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Research Projects Laboratory
George C. Marshall Space Flight Center
Huntsville, Alabama

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BUREAU OF ENGINEERING RESEARCH

Final Report
Contract NAS8-20134
Request No. DCN 1-5-28-01048-01
May 22, 1965 - January 31, 1967

LUNAR RESOURCES: A STUDY OF SURFACE MINING

by

Reynold Q. Shotts, Project Co-Director
Robert M. Cox, Jr., Project Co-Director

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TABLE OF CONTENTS

TABLE OF CONTENTS (Continued)

TABLE OF CONTENTS (Continued)

LIST OF ILLUSTRATIONS

LIST OF ILLUSTRATIONS (Continued)

I

INTRODUCTION

The present report completes the work on lunar mining under the contract. The first year of the contract was spent assembling information on the lunar environment and on the selenology of the lunar surface and shallow sub-surface. Some of the data on the lunar environment have been given conflicting interpretations and for other data, the interpretation is inconclusive. Practically all conclusions reached on selenology are necessarily speculative and are based upon disputed theories of the origin of the moon and of its observable surface feactures. The first report published under the contract 1/, evaluated the data and selected interpretations, or ranges of interpretation, regarding environment. Twelve types of water deposit that might occur within the range of proposed selenologic theories were presented. Since preparation of this report, soft landings of Luna IX and Surveyor I have strengthened some near-consensus opinions but it is doubtful if the ranges of lunar environment and of selenology discussed in the first report have been significantly altered.

The present report deals directly with the problem of winning hydrous ores. There was no time to devote to the problems of ore discovery and deposit evaluation, that normally precede mining, or to ore processing plant or methods, which immediately follow it.

During the first year of the contract, all types of mining systems were reviewed. As a result, it was concluded that the only logical systems for initial consideration were those employing surface, strip or open cast methods. Consequently, the present report is concerned only with surface mining methods or those that require the removal of barren material as large or larger in area, than that of the ore body to be mined. Underground mining methods generally require more labor per unit of ore removed than do surface methods and they are worthy of later study for possible use on the moon, if lunar water deposits prove too deep for surface mining.

In addition to mining systems and their comparative economics, the special topics of rock fragmentation and of power types and sources, applicable to the moon, have been investigated.

1. Acknowledgements: The authors wish to acknowledge, with thanks, representatives of certain manufacturers of mining equipment, electric motors and light metal alloys, who furnished descriptive literature and specifications of equipment. Some of these men took the trouble to write letters giving special data not found in the company literature and suggestions about lunar environmental and mining problems. This kind of help was especially welcomed.

The following is a list of firms and the names of some individuals who provided suggestions or discussed problems:

Allis-Chalmers, Construction Machinery Division, Mr. Larry J. Hill, Product Sales Representative

The Beryllium Corporation

The Brush Beryllium Company

Bucyrus - Erie Company

Caterpillar Tractor Company, Mr. William O. Mooney, Mining Section, Market Division

Esco Corporation, Mr. Fred C. Hahn, Manager of Engineering

Euclid Division of General Motors Corporation, Mr. Edwin C. Fouts, Sales Engineer

General Electric Company, Mr. John M. Greiner, Advanced Analysis, Missile and Space Division

International Harvester Company, Construction Equipment Division

Joy Manufacturing Company

Manitowoc Engineering Company, Mr. Jerome T. Kaminski, International Crane Sales Department

Marion Power Shovel Company, Inc., Mr. J. L. Whaley, Sales Engineer

Melrose Manufacturing Company

Page Engineering Company

Reactive Metals, Inc., Mr. Guenther H. Hille, Manager, Marketing

Joseph T. Rhyerson and Son, Inc.

Titanium Metals Corporation of America, Mr. Robert A. Buddington, Thermal Engineer

Westinghouse Electric Corporation, Mr. F. N. Hines, LAC/DC, Operations Manager, Motor and Gearing Division, and Mr. J. D. Miner, Systems Research and Development Division

II

GENERAL PROBLEMS INVOLVED IN MINING LUNAR WATER DEPOSITS

1. Problems of Selenographic Location: On the earth, mineral deposits often preversely occur in unusual, out-of-the-way and inaccessible areas. In the absence of any clear latitude-or longitude-dependent arrangement of physical features on the moon's surface, we may expect the same thing to occur there. Deductions have been made with regard to possible association of water deposits with types of lunar surface features 1-6/ but none with selenographic position. At the time of the first lunar landing and even after some lunar surface exploration, we may have no particular basis for selecting any given selenographic location over any other, except that of greatest concentration of knowledge. All Ranger and Surveyor landings have been near the lunar equator and although Orbiter I and II pictures have not been seen, the authors assume that most, if not all, the potential landing sites photographed by those two vehicles, were equatorial ones. There is said to be some fuel economy in equatorial over polar landings 7 /, so early landings and probably early bases established for lunar exploration, will be within 10^{o} of the lunar equator. It is practically certain, too, that first landings, first colonies and first mining operations will be on the earth-facing hemisphere of the moon, where direct communication with the earth is possible. Because of superior knowledge we will have of the equatorial areas on the side of the moon facing Earth and the likelihood of first finding water deposits there, it appears equally likely that the first mining operations will be at low lunar latitudes in that hemisphere. There are some effects that these locations will have on mining operations. The most important of these effects is that, for almost the entire lunar day, the deposit will be subjected to the direct rays of the sun. If any part of, or all, the ore consists of free water, it will promptly be evaporated on exposure in the pit, at the high temperatures and hard vacuum existing 1 /.

If lunar materials in the pit walls stand readily, at high angles of repose, there will be a narrow shadow at the foot of the highwall in deeper pits not located right on the equator, for protection from evaporation, provided the active pit wall runs in an east-west direction. If any daytime mining is contemplated, active strip pits will be advanced toward the south, north of the lunar equator and toward the north, south of it. Crater walls, mountainous areas near mare margins, and even the lee side of domes and serpentine ridges, may be taken advantage of, to at least lengthen the period of lunar shadow. Of course, if the ore consists of hydrous minerals, like serpentine, it will be unaffected by exposure to the sun. Other ores such as salt incrustations ($CuSO_4 \cdot 5H_2O$, for example), in which the bound water is in the form of H_2O molecules, rather than hydroxyl radicals, may be partly dehydrated by the direct sun, in the lunar vacuum 5 /. Dangers of free water or loosely bound water evaporation can be avoided by confining mining operations to lunar night-time.

For surface mining operations, moreover, there will be certain additional advantages to night operation, independent of ore composition. Some of these are: (1) practically uniform temperature conditions, even if quite low; (2) lighting from the earth, as earth-shine, much of the time; (3) absence of the solar wind; (4) possible absence or reduction of rarified ionized gases of the thin lunar atmosphere; (5) possible absence or reduction of a postulated surface electric charge; (6) no cooling needed for space suits and/or machine cabs or structures; and (7) better heat dissipation from bearings, motors, etc.

Some possible disadvantages of night operation are: (1) the necessity to provide artificial lighting, at least part of the time; (2) the high heating load for space suits, cabs, and structures, (objectionable only if primary power is solar and battery storage is required for night operation); (3) the increase in freezing of working fluids, if any are used; and (4) the suspension of ore production for half the time.

The latter may not be too objectionable if ore can be stockpiled and processing for water done continuously so as to insure steady water production.

If the ore consists of hydrous minerals requiring large quantities of heat for dehydration, processing plants of sufficient capacity will be designed to process, during the two weeks of light, all the ore mined in two weeks of lunar darkness. This schedule will permit use of solar energy for process heat and thus the mining and processing will complement each other and the same crew can be used for both operations.

2. Problems of Topographic Location: No detailed or extensive study of problems of topographic location of lunar mining operations has been made although topography may greatly influence mining operations. If any of the postulated close associations of mineral deposits with physical features 1-6/ prove correct, only types of topography associated with, or near, the more promising features will have to be specifically considered.

Most of the mineral deposit-surface feature association assumed has been with the margin or surface of the maria, preferably near the edges. The so-called walled plains 8/ or flooded craters, appear to be similar to the maria but on a much smaller scale. The maria are generally quite level and comparatively smooth 9-14/. A mine located on a mare surface should be surrounded by this sort of surface unless near a large crater, rill or fault (Straight Wall). Craters are only about 1/15 as frequent on maria as on upland areas and there are only about 6 percent as many craters per unit area on the maria 15/. Large craters that create extensive rough areas are especially scarce on the maria because all observations indicate a rapid decrease in crater frequency with increase in size, on both maria and terrae 16/.

Rills and fissures in some areas may constitute the most formidable topography. Inside the crater Alphonsus, Ranger 9 photographs 13/ revealed many rills not visible or only obscurely visible through the telescope. Ranger VIII photos also picked up some large, previously inconspicuous rills on Mare Tranquilitis 12/. If there is a veneer of fine soil and rubble over all mare surfaces, as suggested by Surveyor I data 14/, there may be hidden fissures under it, that may influence surface mining or mine transportation. Sec. III, to follow, includes some lunar deposit models that imply possible strong control of mineralization by rills, fissures and shear zones.

The influence of topographic factors on possible lunar resource distribution and on mining problems deserves more intensive and detailed study than could be given it in this project.

3. Lunar Temperature Problems: In a previous report 1/ we have discussed the possible range of, and most probable values for, (1) maximum surface temperature, (2) minimum surface temperature, (3) shallow subsurface temperature, (4) average and maximum proton and α-particle flux, (5) average meteoroid ($>$1gm) and micrometeoroid ($\sim$10 microns) fluxes, (6) sputtering rate, and (7) maximum lunar atmospheric pressure.

The near or actual absence of an atmosphere will require that mining personnel be protected by an artificial, earth-type environment at all times. Such an environment can be secured by (1) a permanent or semi-permanent structure or shelter from which work can be done by remote control, probably involving continuous television monitoring; (2) an environment-conditioned cab mounted on equipment with the work done by mechanical, hydraulic or electronic controls from within the cab and possibly also requiring continuous televiewing; or (3) individual, flexible, environment-conditioned space suits.

With the first two of the three protection systems, personnel performance should be practically independent of atmosphere, pressure, temperature, solar radiation flux and micrometeorite bombardment. Meteorite damage would still be possible but this hazard appears to be small 1 /. Space suits probably can give adequate protection from all these hazards with the possible exception of temperature during the lunar day. Temperatures within a space suit will be difficult to control with full sunlight on one side and shadow on the other. Two possible solutions are (1) conduct mining operations only at night or (2) keep some type of awning over space-suited personnel at all times. The latter solution probably would severely restrict personnel movement but both solutions would provide a low, but uniform, ambient temperature.

Any given temperature, no matter how high or low, can be allowed for in designing mining operations. Temperature differentials and rapidly changing temperatures are, however, more difficult to handle. Results from lunar eclipse measurements indicate that light-shadow temperature contrasts are very large and that changes from light to shadow bring such rapid temperature changes that large mechanical stresses may be set up in metal parts of equipment. Surface temperature changes as great as 174°K/min have been observed 17 / and in metals that are much better heat conductors than the lunar surface, these might exceed 200°K/min. In lunar daylight, parts of a machine may be exposed to sunlight in one spot and in shadow in another. A straight aluminum rod, for example, half in sunlight and half in its own shadow, if a meter long on the underside, may be 1.0375 meters long on top. This would tend to warp the rod appreciably and set up large stresses. The superior heat conductivity of metal tends quickly to relieve sudden stresses induced by passage into or out of shadow.

Surveyor I furnished some information regarding temperature changes at several points on the spacecraft due to shadowing 14 /.

The full effect of such stresses upon working equipment must be taken into consideration in the design and construction of the equipment.

4. Materials and Problems of Lunar Economics: Earth excavating equipment is constructed almost entirely of steel and cast iron. Since transportation of equipment from earth to moon probably will be one of the greatest charges against the utilization of indigeneous lunar sources 18/, some way must be found to reduce weight without impairing the usefulness of the equipment.

Large draglines, and perhaps shovels, make some use of aluminum, especially in booms. One manufacturer writes of a particular model: "We might also add that the 120 foot boom consists of 45 foot steel butt, 38 foot aluminum insert and 37 foot aluminum top" 19/. Thus, 75 feet of the boom length was made of aluminum. The aluminum part weighed 8780 pounds, which is less than equivalent steel parts would weigh. Cabs and other parts normally subjected to comparatively low stresses may also utilize aluminum.

Parts of machines carrying maximum stresses often cannot be made of aluminum or magnesium. Many of these, however, may be replaced by titanium or titanium alloys. Beryllium alloys are also a possibility. One manufacturer 20/ reports yield strengths of 9.9×10^{8} n/m^2 (145,000 psi) for Ti-6A1-4V alloy sheet and plate, 10.07×10^{8} to 11.3×10^{8} n/m^2 (146 to 164,000 psi) for extruded shapes. This compares with about 6.89×10^{8} n/m^2 (100,000 psi) for some heat treated constructional steels 21/. Titanium may not be so desirable with regard to some other properties, but it is said to have superior strength-to-weight ratios, excellent elevated temperature performance, good corrosion resistance and unusual erosion resistance 22/. Nothing was found in manufactuers' data on properties at low temperatures like those of the lunar night but the mean thermal coefficient at expansion (32^{o} - 212^{o}F) is $4.9 \times 10^{-6}/^{o}$F ($8.82 \times 10^{6}/^{o}$K) for Ti-6A1-4V alloy 23/ as compared to about 7 x $10^{-6}/^{o}F$ (12.60 x $10^{-6}/^{o}K$) for 0.2C steel and thus titanium should not be as strongly stressed by rapid temperature changes as is steel.

Densities are approximately 1.74 gm/cm^3 for magnesium; 1.85 gm/cm^3 for beryllium; 2.70 gm/cm^3 for aluminum; 4.5 gm/cm^3 for titanium; 7.85 gm/cm^3 for steel; and 4.43 gm/cm^3 for the alloy, Ti-6A1-4V.

One manufacturer 24/ lists weights of five bulldozers, three crawler loaders and one wheeled loader with the weights of steel used in their construction. The bulldozers averaged 88% steel, the loaders 89.4% and the wheel loader, 91.3%. All steel can hardly be replaced by titanium or aluminum. Titanium, for example, probably does not have the wear resistance of the steel used for bulldozer blade cutting edges although one manufacturer says ". . . titanium has been successfully flame plated with tungsten by the Linde process" 25/, a treatment that should make it much more abrasion-resistant. We estimated weights of earth-designed equipment when (1) Ti-6A1-4V alloy is substituted for much of the steel and (2) electric motors are substituted for diesels. The latter substitution apparently will bring little weight reduction if, to electric motors, must be added heat dissipation radiators. Heat dissipation systems probably will be made largely of aluminum and it is doubtful if these systems for electric motors will weigh as much as the brass cooling radiators of gasoline and diesel motors. If fuel cells or batteries also must be added along with the electric motors, there may even be some increase in weight over diesel engines. Trailing cables will also add weight for that method of electric power distribution.

It must be remembered that lifting, but not necessarily scraping or loading of soil or broken rock, should require only one sixth the power required for the same volume lifted, as on the earth. Thus, power required for equivalent volumetric performance of equipment may well be much less on the moon than on earth. If lunar materials are also of lower density than earth materials, even less power will be required.

5. Materials and Lunar Environmental Problems: In addition to temperature effects, other environmental factors may have direct or indirect short-range or long-range effects upon lunar mining. In an earlier report from this study 1 /, an attempt was made to select limits and most probably values for a large number of factors. In the case of some of them, the affect upon mining is nil or not clear.

A. One of these factors is the lunar vacuum. Its most direct effect will be upon the problems of equipment lubrication since all metals and other common materials of equipment construction probably have too high a vapor pressure to be affected and all fluids will have to be used in sealed systems. At the lunar vacuum, lubricated bearings must be sealed or solid lubricants like graphite, or MoS_2, used. Even solid greases would evaporate rapidly. The combination of low atmospheric pressure, low and high temperatures, rapid temperature changes and no convective cooling (unless built in), will pose some real lubrication problems in design of high-speed equipment for lunar use.

The lunar atmosphere is so rare that it is no factor. There will be no atmospheric oxidation or other chemical effects. If the atmosphere consists of a low temperature plasma of charged Ar and Kr ions, the condition might result in an electrical charge leakage, under some circumstances, but the effect should be small. At earth atmospheric pressure, Ar^+ would be present at a concentration of 2.59×10^{19} ions per cubic centimeter; at lunar atmospheric pressure (maximum possible), this should be less than 2.59×10^{6} ions/cm^3.

The lunar atmosphere gives no shielding from solar radiation or from micrometeoroids.

B. The ultraviolet radiation on the lunar surface, unimpeded by an atmosphere, is much more intense than on earth. In addition, the protons and α-particles of the solar wind, which are repulsed or slowed by our magnetic field and atmosphere, beat full force on the lunar surface. Measures for the adequate protection of personnel from the lunar vacuum should also be sufficient for protection from the solar wind. Protection of equipment and structures from sputtering has been a lively problem but workers on probable lunar sputtering appear to agree that the problem, at most, should be a long term one only and probably will not exist as a practical one for metals or ceramics 26 /.

C. A positive surface electric charge evidently is postulated because of the displacement of electrons by solar wind bombardment. There should be little effect on mining or equipment materials unless more careful grounding or insulation of equipment is required than on earth.

D. The extent of vacuum adhesion is one of the great unanswered questions regarding the lunar surface. Virtually all high vacuum experimental work has demonstrated the presence of adhesion among small silicate particles and between these and some surfaces, including metal surfaces 27, 28 /. None of the studies conclusively prove that in the harder lunar vacuum, dust or larger particles, will settle on and coat surfaces, build up on wheels or tractor treads, or render digging of dust or fine rubble difficult. It is probable that there will be no final answer until a moon landing is made. If vacuum adhesion is a serious problem in lunar mining it may be so only for the first meter, or less, or material. At greater depths, there may be enough water or other condensed volatiles of sufficient vapor pressure for coating nearby or newly created surfaces with vapor so as to prevent the "ultra-clean" condition that may exist at the lunar surface and thus lessen adhesion, even for small particles. An ore containing traces of free water should not be stubbornly adherent.

Surveyor I photographs appear, recently, to have thrown some light on the vacuum adhesion problem. They seem to indicate at least a degree of adhesion between small particles of undoubtedly dry lunar soil about equivalent to that exhibited by damp earth soil 14/. It seems most unlikely that adhesion will prove great enough to interfere seriously with mining in the ways suggested above.

E. Another possible condition, based upon no measurements, is that corrosive gases, vapors, etc., may be encountered beneath the lunar surface. If the moon were melted early in its history, it might be almost completely devolatilized. If heating came later (from radioactivity or from tidal friction), the moon should be only partly devolatilized and many condensible vapors like F_2, Cl_2, HCl, HF, CO_2, SO_2, H_2S, etc. may be entrapped in the cold outer crust. In the case of lunar cold accretional origin, the concentration of condensible vapors (including water) in the outer layers may be low unless they have been driven toward the outside 5/ by internal heating. In case corrosive conditions are encountered, designs of machines for digging, loading, drilling and other uses must incorporate corrosion-resistant materials, and mining and processing system designs must protect personnel and equipment against corrosive and harmful substances.

In free water deposits, ice may be accompanied by other frozen volatiles. Some of these may directly introduce problems of corrosion of mining equipment. For example, if titanium alloys are used in the construction of lightweight equipment and chlorine is present with the ice, severe damage to the titanium parts could occur. Although chlorine attacks titanium very slowly when wet, dry chlorine gas ($<0.5\%$ H_2O) "may react at room temperature" according to one source 23/, while another reference 29/ lists "rapid attack, ignited and burned" ($<0.1\%$ H_2O at 30^oC). In any mixture of chlorine and water, the evaporating chlorine would likely be dry because the boiling point of Cl_2 at 1 atm pressure is -34.6^oC and the vapor pressure of water at that temperature is only a little more than 0.1 mm of mercury.

Some other corrosive substances are bromine, ammonia and steam, calcium chloride, chlorine trifluoride, fluoboric acid, fluorine, flurosilic acid, hydrochloric acid (conc.), hydrofluoric acid, mercury (700^oF), phosphoric acid (boiling, conc.), potassium hydroxide, sodium bifluoride and sulfuric acid ($>$ room temperature) 23 /. Most of these substances attack titanium only at moderate rates. In general, titanium and its alloys may be attacked if the lunar chemistry proves to be strongly reducing. "Titanium provides excellent resistance to general and localized attack under most oxidizing, neutral and inhibited reducing conditions. It also remains passive under mildly reducing condition, although it may be attacked by strongly reducing or complexing media" 23 /.

Aluminum, beryllium and magnesium may have similar reactions with possible water-associated volatiles. Electrochemical reactions capable of inducing corrosion of the metal parts of mining equipment, are also possible in cases of continuous contact. The general chemical reactivity of aluminum and magnesium are well known. The rarer beryllium appears also to be reactive to halogens and some other at earth-surface and higher temperatures 30 /.

The problem of chemical properties of possible substitute metals and alloys in the construction of surface mining equipment was not gone into extensively in this study. Nothing, for example, was found on these properties at lunar night temperatures of $\sim$100-150^{o}K. Such a study might show that, at these temperatures, possible deleterious chemical and electrochemical reactions will proceed so slowly that this factor may be removed from the realm of practical problems in lunar surface mining.

6. Some General Equipment Problems: Many problems will be encountered in the selection and design of lunar mining equipment. Just as the first automobile resembled a buggy, so it is quite likely that the first lunar mining equipment will be earth-models, modified for the lunar environment and mining conditions. This is one assumption on which the present work on mining systems is based. After some lunar mining experience, undoubtedly original and different equipment designs will be developed. If lunar mining is preceded by a considerable period of lunar exploration, there will be opportunity for new designs to be applied to transportation vehicles

and methods and to drilling techniques. Any underground shelter construction activities, prior to mining, would also give some experience in problems of fragmentation, digging, loading and limited-scale excavation. These activities may, however, resemble underground mining more than surface methods.

Problems briefly considered to date are: (1) power supply (see Section V); (2) mode of traction for moving equipment; (3) equipment control methods; and, (4) some secondary or service materials, such as for ballast or counterweights, maintenance facilities, roads, bins, crusher foundations, control systems and power distribution systems.

A. Present small excavation equipment moves on treads or on wheels, usually rubber-tired. Larger shovels and draglines are crawlers (treads) or walkers. Mitchum 31 / concluded that tracked vehicles are less efficient than wheeled ones. They are heavier and less reliable and their only superiority, which is slight, is in soft, dry granular material. In view of (1) Surveyor I photo indications of the probable presence of such material 14/, (2) wheels must be rather large to cross cracks easily negotiated by equal-sized crawler vehicles and (3) the questionable practicability of lunar use of pneumatic tires, it is assumed that tracked vehicles will be used for mining equipment.

B. Mining equipment may conceivably be controlled by: (1) an operator in a space suit, much as in operations on earth, a method probably practical only for lunar night work; (2) an operator in an enclosed, earth-environment cab mounted on each piece of equipment, whether bulldozer, loader, shovel, or dragline; (3) remote, automatic controls from an earth-environment shelter especially erected at the mine or from the crew living shelter which may or may not be located at the mine; or (4) remote, automatic control from the earth. The last two control methods probably will require continuous television viewing.

At least one mining system using central control at the mine and a number of others using either of the first two methods will be considered. The necessity to study the first two in detail has been eliminated by limiting our study to lunar night work. Neglected also are really "remote" control systems.

It is readily acknowledged that systems (1) and (2) will use more lunar-based labor than will the others and that this large cost item may constitute its greatest handicap.

C. The category of secondary or service materials is limited to consideration of two problems: The first of these is based on the fact that there should be on or near the lunar surface a considerable accumulation of meteoritic nickel-iron. Whether it occurs in discrete large or small masses, or as condensed vapor from the heat of impact, dispersed throughout the lunar soil, is not known. If it proves collectable, it should be ideal for use as ballast or counterweight in construction equipment needing such material. A given volume of it, of equivalent density, should be six times as effective for this use as the same volume would be on earth. It would be most uneconomic to ship ballast to the moon but at the time of equipment erection on the lunar surface, ballast or counterweights may be provided locally from meteoritic iron. If counterweights must be monolitic, small meteorite iron masses embedded in concrete would be ideal. Green 32/ has suggested cast basalt or sulfur as possible concrete substitutes. If mining operations are limited to night, the latter should be satisfactory (m.p. 112.8°C).

The second suggestion on secondary materials is compelled by the necessity to provide an ore chute or bin as the minimum auxiliary facility at a mine. If mining and haulage can be combined in one operation, such as with a scraper-loader, the bin or chute could be placed at the processing plant and ore hauled there directly. If ore is loaded into vehicles (trucks) by a front-end loader, shovel or dragline, or if it is merely stockpiled at the mine and loaded and shipped later, no chute or bin may be required. As will be seen later, however, any scraper mining system or any automatic control system, will require some kind of chutes or bins. If the ore is to be crushed at the mine, this will require at least a chute and a foundation. As will be shown in illustrations following, it is proposed that metal posts be used for such structures. In earth soil, these could be placed by a piledriver; this would be possible also in the lunar soil shown in Surveyor I photographs provided (a) it is deep enough and (b) there are no large boulders, meteorites, iron masses, etc., to prevent driving. In the case of this type of condition and of solid rock, posts could still be placed in holes drilled for them.

7. Mining Schedule: One published schedule suggests a lunar oxygen demand of 4,546 kg/month (10,000 lb/month) or 54,552 kg/year (120,000 lb/year) by 1976 and double this quantity in 1982 18/. The 1982 quantity will be used for estimates. It is equivalent to 122,728 kg (270,000 lb) of water per year. On this basis, ore averaging 2 percent water, by weight, must be mined and processed at the rate of 6,136,367 kg (13,500,000 lb) per year, if extraction is 100 percent efficient. We shall arbitrarily assume 6,818,200 kg (15,000,000 lb or 7500 tons) as the basic annual quantity to be mined from either of the above deposits, or an implied over-all recovery of 90 percent.

A. Mining will be assumed to be limited to one 8-hour shift per 24 hours and to lunar night only. The lunar night is equal to 13.66 earth days and there are 13.37 lunar nights per earth year. This gives 182.63 shifts per year. For simplification of calculations, 180 shifts per year will be used. If mining is also done during the lunar day, it may be expanded to 360 shifts.

B. For 180 shifts, the mining rate will be 37,879 kg (83,333 lbs) ore per shift.

C. The volume of ore produced to get the required mass will be highly variable. In Section III, the seven hypothetical mineral deposits have ores of average density ranging from 1200 kg/m^3 (75 lb/ft^3) to 3000 kg/m^3 (187 lb/ft^3). Volume requirements will therefore be from 31.57 m^3 (41.29 yd^3) to 12.63 m^3 (16.52 yd^3), per shift.

D. Overburden or wall rock that must be excavated also varies in density from 1250 kg/m^3 (78 lb/ft^3) to 2500 kg/m^3 (156 lb/ft^3) and from 30.30 m^3 (39.57 yd^3) to 15.15 m^3 (19.78 yd^3) must be mined, per shift, during shifts spent exclusively on overburden excavation.

III

SELECTED HYPOTHETICAL MINERAL DEPOSITS

Just as fundamental as lunar surface and environmental problems are those of the kind, grade, extent, structure, and location of the deposits that may contain water or other useful minerals. At this point, of course, we have no direct evidence of the presence of mineral deposits at all. The presence of water or any other mineral deposit can only be inferred and inferences must, in turn, rest upon assumptions regarding lunar origin, thermal history, chemical composition, and the extent of near-surface chemical differentiation. No attempt will be made to explore here the many possibilities that have been suggested.

At the present time, when considering mining problems and techniques, it will be necessary to cover the entire range of possible chemical differentiation of the lunar crust. In a previous report 1/ twelve deposit models were examined. They ranged from strictly impact origin to both intrusive and extrusive volcanic models. Ten of them were taken from the work of Salisbury 4, 33/ and of Westhusing and Crowe 6/. Nine of the twelve models are associated with the lunar maria. Of the nine, four imply the maria consist of bedded tuff layers, three that they are filled with successive lava flows, and two could be interpreted as a filling from one thick lava outpouring. One lunar upland model consists of a single rubble blanket over granodiorite, a second, of interbedded, lenticular rubble layers over granodiorite and the third is a rubble layer alone, with imbedded serpentine boulders.

If lunar mineral deposits occur in the form of local mineral enrichments, as do earth deposits, a program for their discovery, evaluation and exploitation will be necessary. If they can be associated early with surface features (maria edges, crater bottoms, crater rims, lunar domes) or with visible or hidden structural features (fracture systems, fluid vents, etc.) or if they are found to occur in groups or geographic districts, as on earth, the problem of discovery and evaluation will be greatly simplified. Unless concentration does occur to some extent, drilling or other exploration to evaluate the deposit will be useless and mining will become merely the random removal of country rock.

Water occurring in free form should be more economical to mine and process than chemically combined water. If free water occupies pores, shear zones or small fractures in hard or tough rock, mining it will be almost as difficult as for hydrous minerals in similar rock. Permafrost zones in tuff, and similar free water deposits, should be much less difficult to mine.

Seven hypothetical models have been selected for quantitative description. Deposit geometry and dimensions have been assigned. Calculations have been made of the volume and, in some cases, the mass of ore and overburden that must be removed to supply a fixed annual demand for water on the moon. Any mining problem peculiar to the model assumed, is discussed.

The deposit model and its quantitative evaluation merely serves as a constant basis on which to compare alternate mining systems equipment, but an attempt has been made to make the assumptions as realistic and consistent with the present state of our knowledge and speculative extrapolation as possible.

1. Hypothetical Deposit No. 1: A. This deposit is a uniform permafrost zone, 3m (9.84 ft) thick, in a friable volcanic tuff.

The zone lies at a uniform depth of 10 meters below a generally level mare surface, similar to Surveyor I terrain 14/. Overburden is a dry, "semi-welded" tuff.

Fig. 1 is a section of the deposit. It is essentially Model 1 of Westhusing and Crowe 6/.

B. The ore has a density of 1200 kg/m^3 (75 lb/ft^3) and the overburden, 1250 kg/m^3 (78 lb/ft^3).

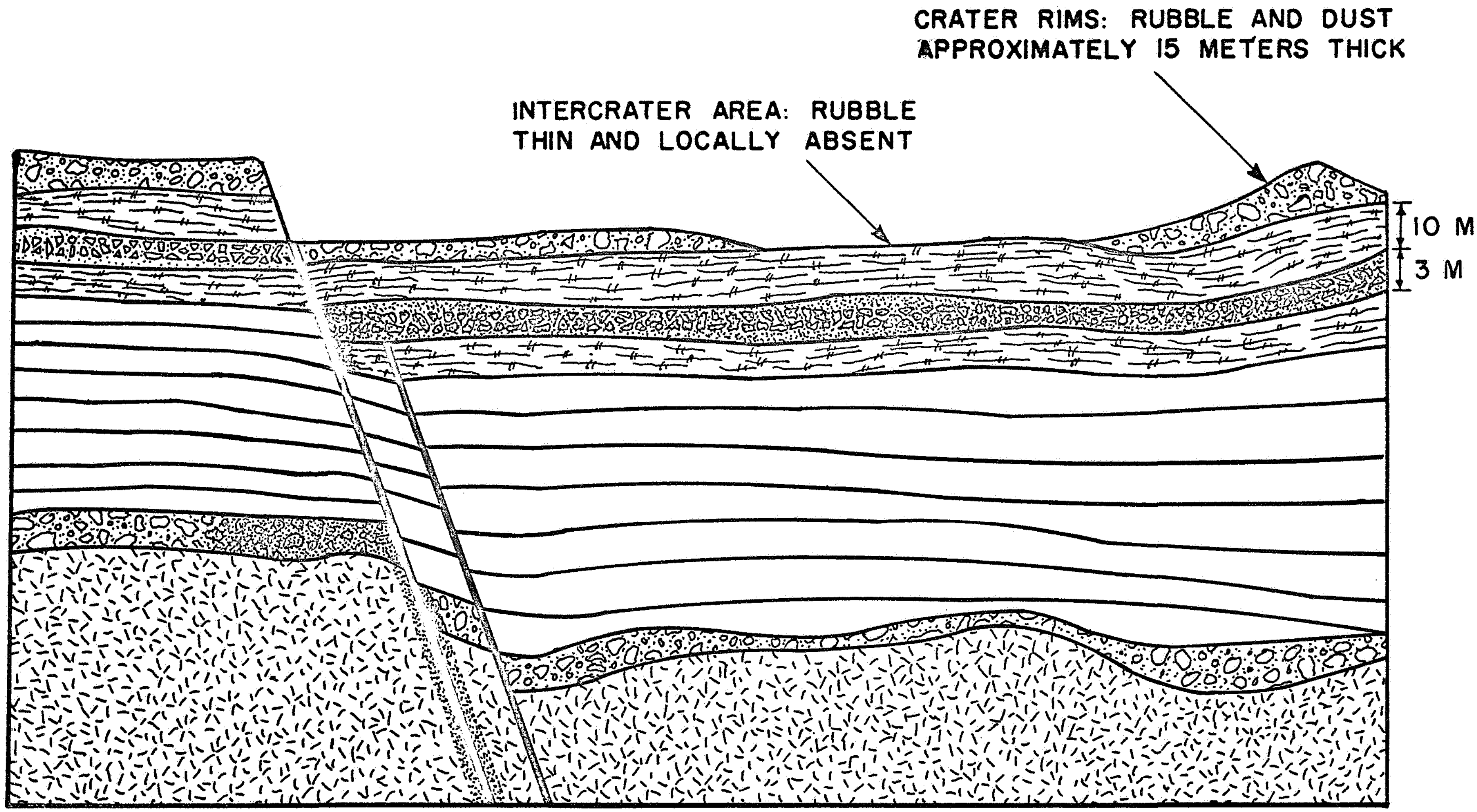
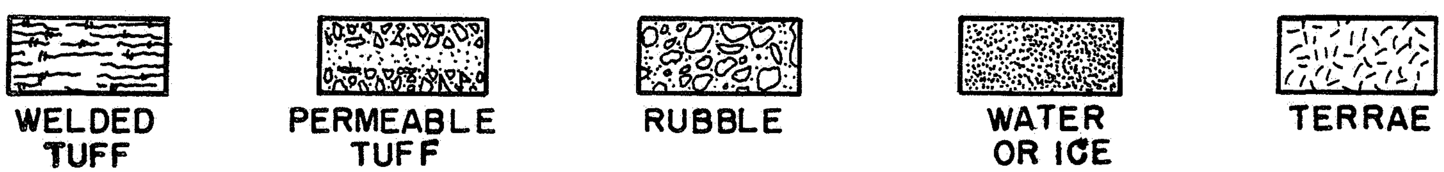


Fig. 1. Section across hypothetical deposit No. 1, a permafrost zone with shallow cover.



The required mass of 37,879 kg (83,333 lb) ore per shift, makes a volume of ore to be mined, of 31.57m^3 (41.29 yd^3), per shift.

The ratio of overburden to ore in the deposit is 10/3 and the overburden that must be moved will be 105.23m^3 or a total of 136.80m^3 (178.93 yd^3) of material per shift.

C. Mining of this deposit by conventional open cast or stripping methods will resemble very closely, area strip mining of coal or any other bedded mineral deposit. Mining will be started by a box cut and the overburden piled on the edge of the deposit if mining can begin near an edge, or mare border. After removal of the ore in this cut, overburden from each successive cut can be piled in the area from which ore has just been removed.

If a pit width of 25m (82 ft) is arbitrarily assumed, the distance that the pit must be advanced, each shift, to remove 31.57m^3 of ore, or a block 3m high and 25m wide, will be only 0.42m (1.38 ft). In a year (180 shifts) this will be only 75.6m (248 ft) or about 3 times the pit width. With such a low rate of pit lengthening, haulage distance will increase quite slowly. In a relatively wide pit, stationary scraper systems will require infrequent shifting of head and tail towers or pulleys.

Both ore and overburden can be scraped or dug without prior fragmentation.

2. Hypothetical Deposit No. 2: A. This deposit is identical to No. 1, except that the overburden is 30m (98.4 ft) thick. See Fig. 2.

B. The overburden-to-ore ratio in this deposit is 10/1 and its mining will require removal of 31.57 m^3 (41.29 yd^3) of ore and 315.7 m^3 of overburden or a total of 347.3 m^3 (454.2 yd^3) of material per shift.

C. This deposit can be mined by the same methods as deposit No. 1 but will require larger equipment. The overburden probably must be mined in two or more benches. Disposal of the overburden may prove a formidable task, especially if (1) its angle of repose proves low or (2) its bulk density is so reduced when loosened

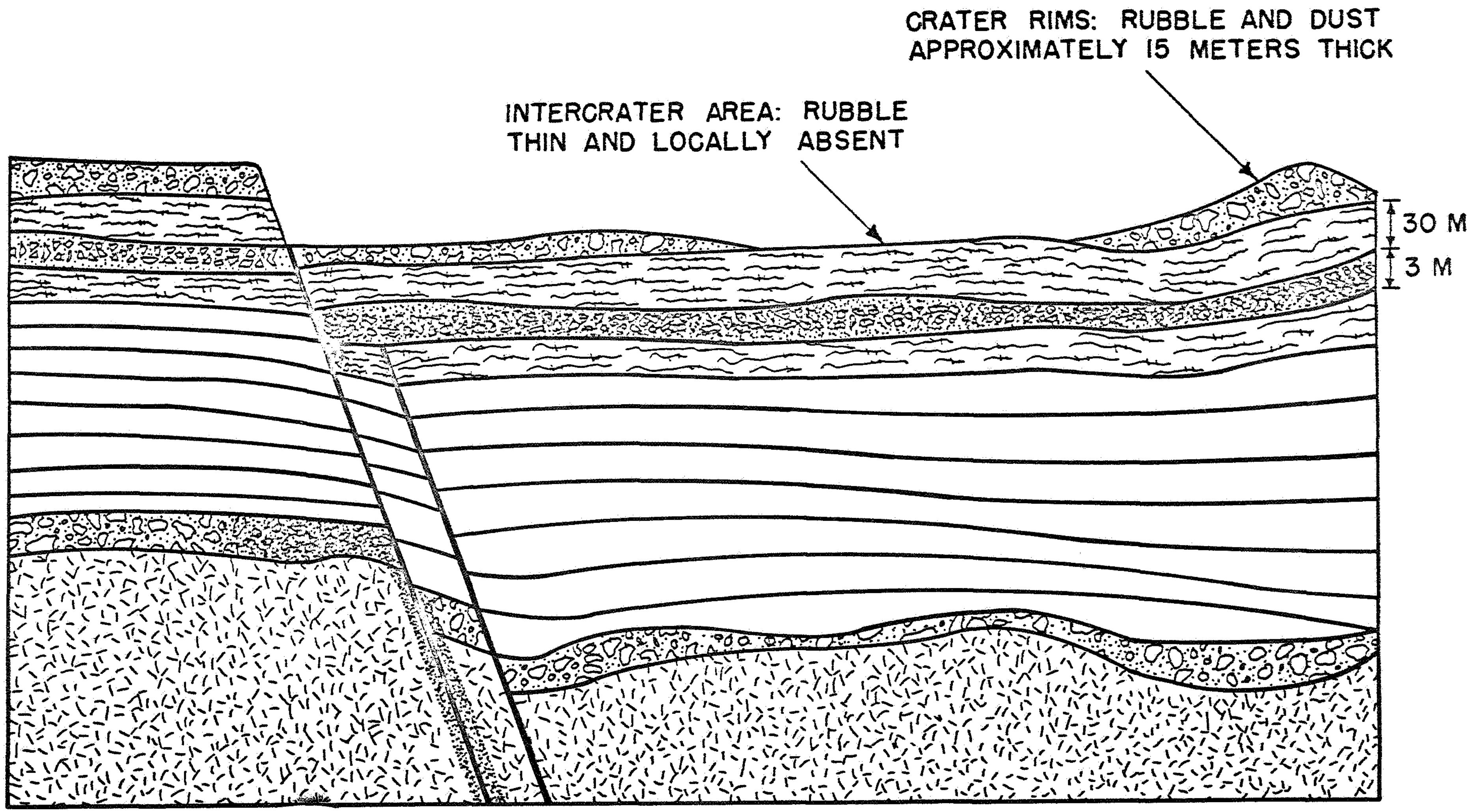
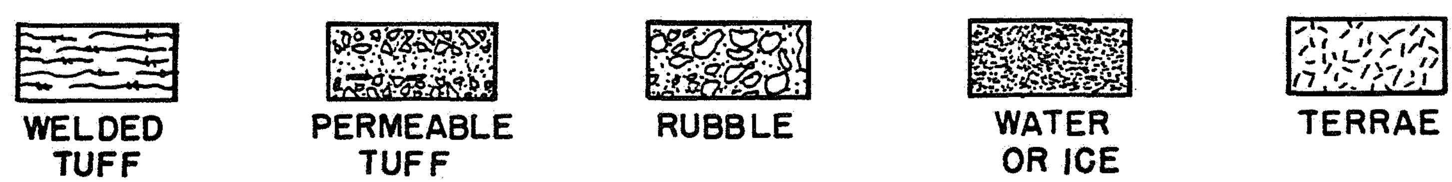


Fig. 2. Section across hypothetical deposit No. 2, a permafrost zone with thick cover.



by digging that all of it cannot be piled in the pit left on ore removal. In either case, it may be necessary to haul some of the overburden a considerable distance and thus add to the cost of mining.

3. Hypothetical Deposit No. 3: A. This deposit is an intensely fractured and mineralized zone under an upland crater. The dimensions are as shown in Fig. 3.

In Fig. 3 it can be seen that the ore deposit is approximately a spherical segment of radius 50 m and height 30 m. The volume of a spherical segment = $1/6\pi h$ $(h^2 + a^2)$ where a is the radius of the base of the segment, or 50 m. The volume is $0.5326 \times 30\ [(30)^2 + 3(50)^2] = 134,232\ m^3$.

Overburden depth is more complex. In the center of the crater it is 1 m. The outer slope of the rim is from 0 to 6 m in thickness and its horizontal width is 25 m. The inner slope decreases from 6 to 1 m in 15 m distance. This gives the outer slope an inclination of $\tan^{-1} 16/25 = 0.295$ or 16°26' and the inner slope, $\tan^{-1} 5/15 = 0.333$ or 18°26'. This can be idealized as (1) a disc of trapezoidal crossection, 1 m thick and 50 m radius (because of the tapering, its volume is a little less than calculated as a flat cylinder), and (2) an annular ring of equilateral triangular crossection, 5 m high, 30 m base and with an outer radius of 42 m and an inner one of 10 m. The disc has a volume of (almost) $\pi r^2 h = \pi(50)^2 \times 1$ or 7854 m^3. The volume of the triangular section should be that of half a hollow cylinder of outer radius 42 m, inner radius 10 m and 5 m height or vol. = $\pi/2 \times h\ (r_1^2 - r_2^2) =$ 13,069 m^3. Thus, total volume of overburden to be moved = 20,923 m^3 and of ore and overburden together 155,155 m^3. The ratio, overburden/ore is 0.16 or the overburden constitutes only 13.5 percent of the material to be moved.

B. The ore is assumed to be free water released by the heat of impact and redeposited in the sub-crater breccia as the ground cooled. The ore could also be a hydrated (serpentinized) outer film or zone on breccia blocks with small particles completely hydrated. Ore will be assumed to average two percent water.

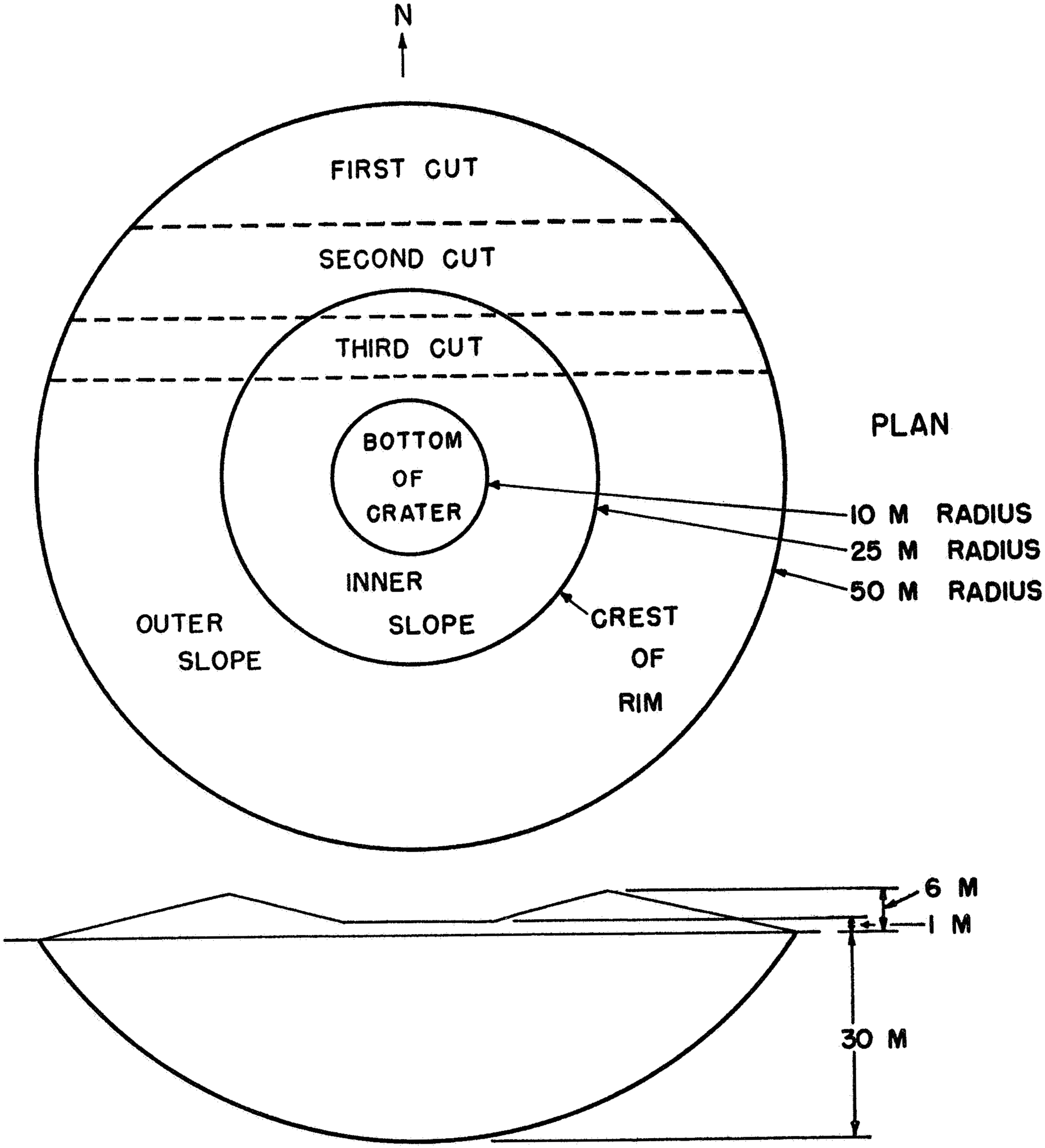


Fig. 3. Section across hypothetical deposit No. 3, a fractured and mineralized zone under an upland crater.

If the general granodiorite model of Westhusing and Crowe 6 / is chosen, the solid rock has a density of about 2.67 gm/cm^3 34/ but the breccia will be assumed to have an average bulk density of about 2400 kg/m^3 (150 lb/ft^3). The breccia will be assumed blocky with maximum block size of about 10 cm.

Overburden will be assumed to be rubble (fine within crater; coarser on crater rim) with stray blocks as large as 1 meter 14 /. Rubble density is assumed to be 2000 kg/m^3 (135 lb/ft^3).

If 1 m^3 ore weighs 2400 kg and is 2% water, each m^3 ore contains 48 kg water. In 134,232 m^3 ore, there are 6,443,000 kg H_2O. At a consumption rate of 122,728 kg water per year 18 / and assuming 90% water recovery, ore to produce 136,360 kg water per year, will be needed. This rather small sub-crater deposit, at this rate, will have a life of 47.2 years. A one percent water deposit of this size would last 23.6 years. A deposit of less than one percent water would not be an ore on earth and hardly distinguishable from average rock 2 /.

With only 155,155 m^3 total material, the quantity to handle will be 3287 m^3/year or 3287/180 = 18.26 m^3 (23.88 yd^3) per shift. This volume is less than for either deposit No. 1 or No. 2, because (1) the ore is denser and therefore has a larger volumetric water content at two percent by weight, and (2) the overburden-to-ore ratio is much lower.

The mass of material to be handled will average 42,836 kg (47.2 tons) per shift.

C. The mining rate on this deposit would not be quite uniform. As the deposit is flush with the surface at the toe of the crater wall and thickens toward the center, mining will begin at the toe of the wall away from the lunar equator and move, keeping an east-west working face, toward the center. This procedure will keep the working face in shadow during the lunar day, a condition that will not only protect the ore from too rapid daytime evaporation, if all or part is free water, but will also make a little daytime mining possible, if the deposit is not too close to the lunar equator.

This mining pattern, for a uniform water production rate, will require rapid advance of the face at first, where the ore lens is thin, with each successive slice either narrower, as shown, or with the face advancing more slowly, as the center is approached. When half the deposit is mined, the ore body will thin and advance of the face must be more rapid. For the first 25 m into the lens, the overburden mined will increase to a maximum, then decrease. After 75 m of advance, the decrease in overburden removed will be steady.

It will be assumed that the overburden is scrapable but the ore (1) must be dug or (2), if cemented with ice of other material, must be drilled and broken before loading.

4. Hypothetical Deposit No. 4: A. This deposit is a complex, mineralized fracture or shear zone in basaltic lava in but near the margin of a mare, as shown in Fig. 4. The zone is 100 m wide and 300 m long. The top 3 meters of the zone is

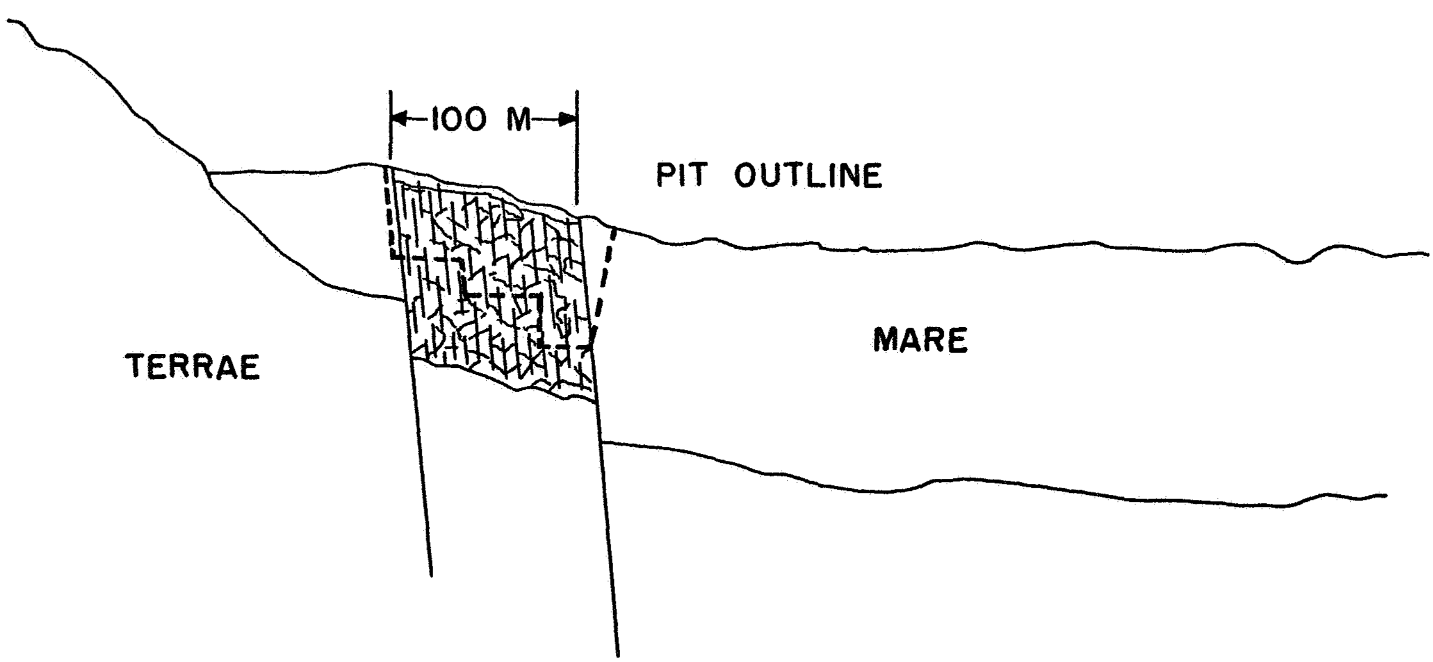


Fig. 4. Section across hypothetical deposit No. 4, a mineralized fracture or shear zone near the margin of a mare.

barren rubble. Mining may be continued to the bottom of the mare but an arbitrary cut-off zone at 103 m, will be assumed. The volume of ore present is simply 100x100x300 m or $3x10^6$ m^3.

B. The ore will be assumed to be free water or hydrous or serpentenized surfaces breccia blocks within the shear zone to yield an average of 2 percent water. Ore density will be assumed to be 2500 kg/m^3 (156 lb/ft^3) and the rubble, 2000 kg/m^3 (135 lb/ft^3).

The deposit is also a large one, containing $150x10^6$ kg of water. At the assumed 1982 water demand and 90% recovery, it should last $150x10^6$/136,360 = 1100 years.

Because the shear zone is not quite vertical, extra material must be removed on the hanging wall side, as shown in Fig. 4. This condition probably would dictate the mining be shallow in so large a deposit in order to avoid moving extra gangue. Fig. 4 suggests that the lava stands well but if it does not, some rock may also have to be removed on the foot wall side. In Fig. 4, it appears that about 700 m^3 of country rock must be removed per meter of length of the zone at the 70 m (230 ft) depth shown or of about 1300 m^3 at 100 m depth.

C. Mining can be quite flexible. With only a little overburden to move, it can be disposed of in almost any way. Mining can be deep or shallow. The ore is assumed to have only a barely preceptable tendency for preferential breakage along the shear surfaces and must be fragmented and dug.

Reducing this ore to one percent water would still make it a very large deposit for the assumed 1982 water demand level.

5. Hypothetical Deposit No. 5: A. Fig. 5 indicates that deposit No. 5 is a rill filled by rubble and probably cemented by mineralization. A similar model is possible for intruded lava. An intruded lava ore should be denser than mineralized rubble.

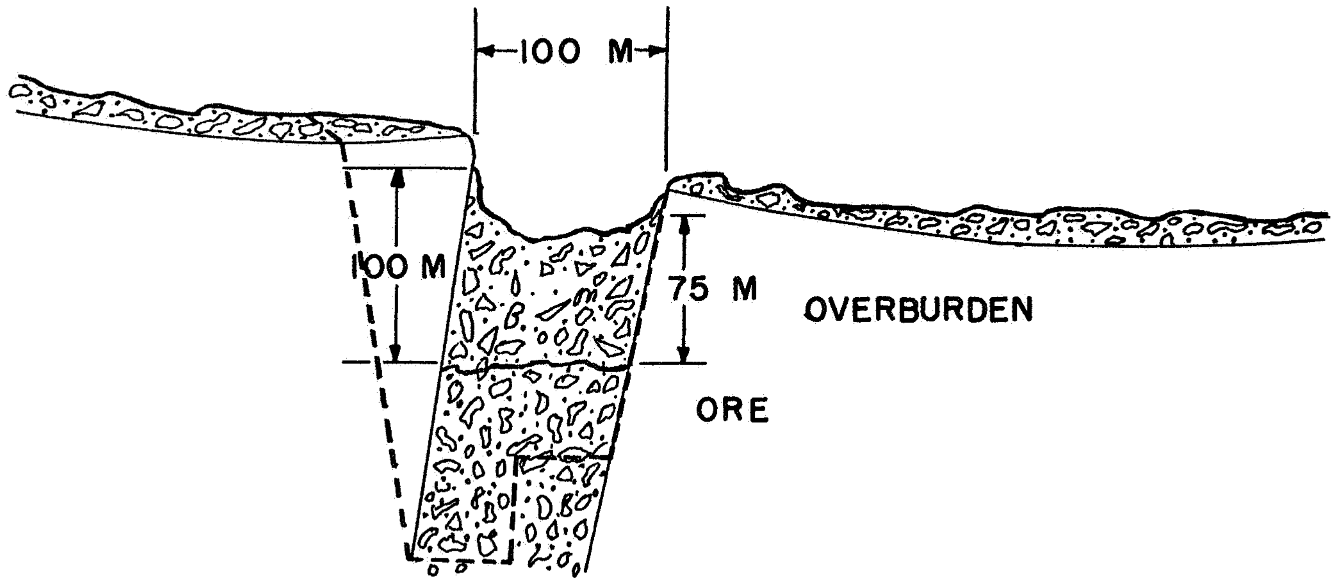


Fig. 5. Section across hypothetical deposit No. 5, a rubble-filled rill on a mare surface.

Dimensions shown in Fig. 5 are a rill width of 100 m and a depth of 100 m to the ore zone, on the footwall side. The length of the rill may be assumed to be 300 m. If mining is possible to a depth of 100 m of ore, this deposit has similar geometry to deposit No. 4 but (1) it has an average of 75 m depth of barren rubble to be moved to reach the ore and (2) its greater depth and lesser dip will require removal of more barren rock on the hanging wall side. Total volume of the deposit is 100x100x300 = $3x10^6$ m^3, less a small quantity due to a slight narrowing of the rill, with depth, as indicated.

B. The ore may be free interstitial water or hydrous (serpentized) rubble or both. Density of the ore will be assumed as 1200 kg/m^3 (75 lb/ft^3) and the unmineralized rubble, as 1250 kg/m^3 (78 lb/ft^3). The enclosing lava may be slightly distended and have a density of about 2500 kg/m^3 (156 lb/ft^3). The volume of overburden to be removed to mine all the ore in the deposit, up to 100 m of ore, is 75x100x300 = $2.25x10^6/m^3$. This gives a ratio of 0.75m^3overburden per m^3 ore. If less than 100 m depth of ore is mined, this ratio goes up.

The profile shown in Fig. 5 gives an additional 2.7×10^6 m^3, approximately, of rock that must be removed. This raises barren material to 4.95×10^6 m^3 and the ratio of barren material to ore is 1.65. Tonnage-wise, the ratio is even larger because the 2.7×10^6 m^3 of rock of density 2400 kg/m^3 or 156 lb/ft^3 is denser than either rubble or ore.

If each m^3 of ore weighs 1200 kg (2646 lb) and is two percent water, each m^3 ore contains 24 kg (52.9 lb) water. In the entire deposit there will be $3 \times 10^6 \times 24$ = 72×10^6 kg water (79,400 tons). At the assumed 1982 water demand and 90% recovery, the deposit would last $72 \times 10^6/136,360$ = 528 years and thus is also a very large one.

To mine the entire deposit requires removal of 3×10^6 m^3 ore, + 2.25×10^6 m^3 barren rubble and 2.7×10^6 m^3 of hanging wall rock, or a total of 7.95×10^6 m^3 of material. To mine 136,360 kg water per year requires removal of 19,694 yd^3 = 5,682 m^3 (7,432 yd^3) of ore or 15,057 m^3 (19,694 yd^3) of material. This averages 83.7 m^3 (109.5 yd^3) per shift. This material divides into 31.6 m^3 (41.3 yd^3) ore, 23.7 m^3 (31.0 yd^3) barren rubble and 28.4 m^3 (37.1 yd^3) mare rock. Weights of these materials are respectively 37,920 kg (41.8 tons), 29,625 kg (32.7 tons), and 71,000 kg (78.1 tons) for a total of 138,545 kg (152.7 tons).

C. Mining this deposit will require the removal of much overburden before any ore is reached. Skimming the top of the deposit, rather than digging to a physically maximum depth, increases cost per ton of ore recovered. In planning the mining of this deposit, method and equipment capable of deep mining, undoubtedly will be chosen.

As a rill bottom is depressed below the surrounding surface, access to it probably can be constructed on a natural ramp (slide material) or on maria material, scraped or blasted into the rill at a chosen point. The first cut made to the deposit will necessitate the removal of much overburden which must be piled outside the rill, if this is possible, but if not, it will be piled in the rill and handled again, when that portion of the rill is mined. If the first cut is across the rill, for every meter of advance along it, $7500 m^3$ of overburden must be removed. Thus, in a box cut, 50 m long and across the full width of the rill, 37.5×10^4 m^3 of material must be removed to uncover 0.5×10^4 m^2 of the deposit, not including the mare material that must be removed on the hanging wall. This is estimated, from Fig. 5, as about 32×10^4 m^3, making a total of 67.5×10^4 m^3 of material to mine per m of advance.

6. Hypothetical Deposit No. 6: A. This deposit is a lunar dome, assumed to be a serpentine laccolith. Fig. 6 indicates the geometry and dimensions of the deposit. The deposit is essentially Model 4 of Westhusing and Crowe 6 /. It will be assumed circular in plan (radius 500m) and its volume will be treated as a circular segment of height 100 m. Overburden is anhydrous tuff of density, 1250 kg/m^3 (78 lb/ft^3) and is 100 m thick. The upper one or two meters will be fine-grained 14/.

The ore will be assumed to contain two percent water. A pure serpentine may run as high as 12-17 percent water 35/ which may be taken as the upper limit of water content for any individual particles in the deposit. For a water content of 2 percent, the bulk of the ore will be constituted of basic igneous minerals that are serpentinized.

Wahlstrom 36/ lists the following density ranges for basic rocks and serpentine minerals:

olivine: 3.3-4.3 gm/cm^3

orthopyroxine: 3.28-3.94 gm/cm^3

diopside-hedenbergite: 3.26-3.46 in.

hornblende: 3.0-3.5 gm/cm^3

pigeonite: $3.4\pm$ gm/cm^3

chrysotile: 2.36-2.5 in.

antigorite: $2.5\pm$ in.

It is evident that the serpentines, antiogrite and chrysotile, are of lower density than the basic rocks from which they may have been metamorphased. With only 2 percent water, more parent rock than serpentine would be present and a density of 3000 kg/m^3 (187 lb/ft^3) will be assumed.

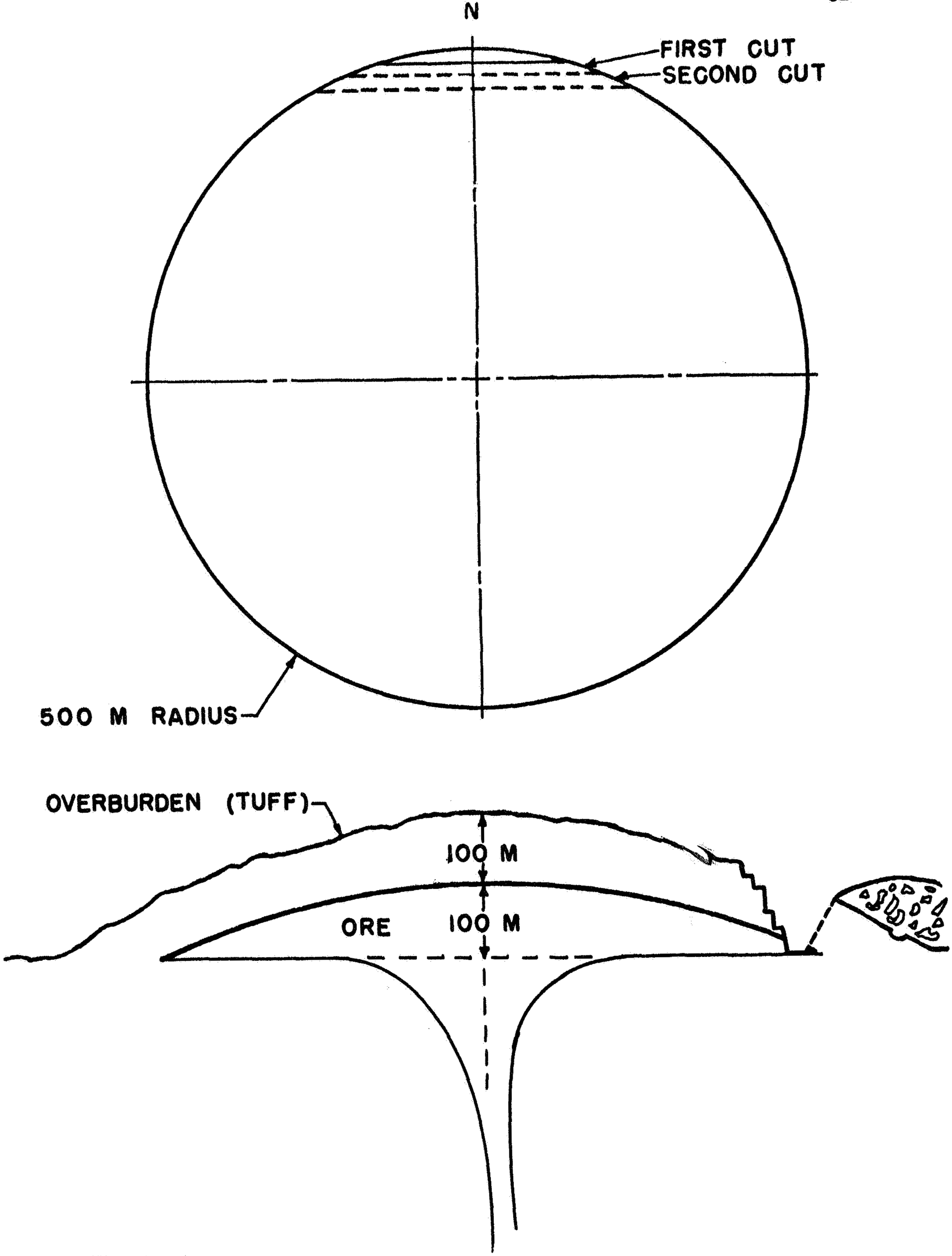


Fig. 6. Section across hypothetical deposit No. 6, a serpentine laccolith on a mare (dome).

The volume of the mineral deposit will be: $1/6\pi h\,(h^2 + 3a^2) = 0.5236 \times 100 [(100)^2 + 3 \times (500)^2] = 39.8 \times 10^6\ m^3$ ($41.3 \times 10^6\ yd^3$). At an average density of 3000 kg/m^3, this gives a mass of 119.4×10^9 kg and a water content of $119.4 \times 10^9 \times 2 \times 10^{-2} = 238.8 \times 10^7$ kg (31,240 tons). At the 1982 water demand and 90 percent water recovery the life of the deposit will be $2.388 \times 10^9 / 1.3636 \times 10^5 = 17,500$ years, another very large deposit.

Overburden volume is some more than $\pi r^2 h = 3.1416 \times (500)^2 \times 100 = 78.5 \times 10^6\ m^3$ because of the domed surface of the deposit. For the entire deposit, the overburden to ore ratio is 1.97/1 and the volume of ore constitutes only 34 percent of the volume of material to be mined.

C. Because of the uniform depth of overburden, mining can begin at any place on the deposit. If it is begun on the edge, little ore will be recovered in the first cut. For this reason it is probable that mining will begin far enough from the edge to give a substantial depth of ore. If we assume this depth is 3m (9.28 ft), the distance from the edge will be no more than 50 m because of the rapid thickening of the segment. The 50 m or so of ore in the berm will probably be left, in a large deposit like this one. Fig. 6 shows that the box cut slopes on both sides. The degree of slope will be a function of the ability of the overburden (tuff) to stand. Material of any given internal strength should stand on the moon, with a steeper angle of repose than will material of the same strength in the earth's 6x gravitational field. In actual practice, the slopes probably will be stepped as shown in Fig. 6, for the inside slope.

If the overburden has little internal adhesion, it can be scraped and loaded directly. The ore will require fragmentation prior to loading. Fragmentation, on the small scale of mining assumed for 1982, probably will be by the drilling, blasting and loading cycle, probably with chemical explosives, as is common on earth.

7. <u>Hypothetical Deposit No. 7</u>: A. This hypothetical deposit was added to six original ones because it appeared to be one that might successfully be mined by a ditching machine or a hoe.

The deposit is a variation or hybrid of deposits nos. 4 and 5 and consists of parallel, mineralized faults near the edge of a mare, averaging 1.55m (5 ft) wide with mineralization 2m (6.56 ft) below the surface and extending downward indefinitely. For simplicity it will be assumed that the cracks are filled even with the surface of the mare and are spaced about 25 m apart as shown in Fig. 7. The cracks can be filled with rubble, lava, or incrusted, porous salts (some hydrous), sulfur, mercury, etc., brought to the surface by water and other volatiles as the lava cooled in a mare of "flooded" crater. For this model, the salts, sulfur, etc., will be assumed the source "ore".

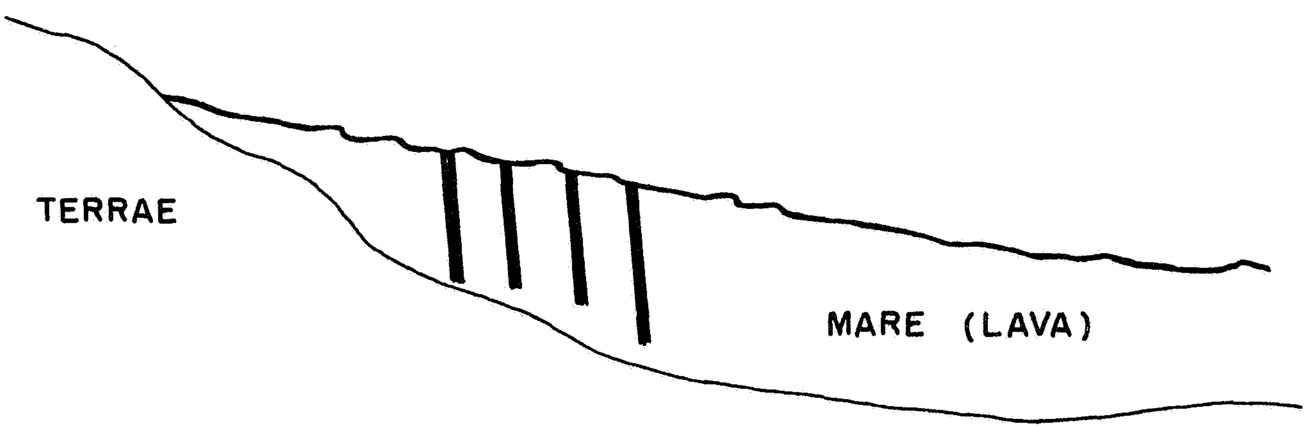


Fig. 7. Section across hypothetical deposit No. 6, parallel, narrow, mineralized fractures near the edge of a mare.

As these ore bodies are too narrow to be surface-mined by most common strip mining equipment without too much overburden moving, the ore "reserves" will depend upon the possible mining depth with a ditching wheel or hoe. If the cracks average 300 m long, one will have the following dimensions:

Depth below surface	Overburden Volume	Ore Volume
2m (6.56 ft)	900 m^3 (1177 yd^3)	0 m^3 (0 yd^3)
3m (9.84 ft)	" "	450 m^3 (5886 yd^3)
4m (13.12 ft)	" "	900 m^3 (1177 yd^3)
5m (16.40 ft)	" "	1350 m^3 (1766 yd^3)
6m (19.68 ft)	" "	1800 m^3 (2354 yd^3)
7m (22.96 ft)	" "	2250 m^3 (2943 yd^3)
8m (26.24 ft)	" "	2700 m^3 (3531 yd^3)
9m (29.52 ft)	" "	3150 m^3 (4120 yd^3)
10m (32.80 ft)	" "	3600 m^3 (4709 yd^3)
12m (39.36 ft)	" "	4500 m^3 (5886 yd^3)
15m (49.20 ft)	" "	5850 m^3 (7651 yd^3)
20m (65.60 ft)	" "	8100 m^3 (10,594 yd^3)

B. The average density of the ore is 1200 kg/m^3 (75 lb/ft^3) and of the overburden, 1250 kg/m^3 (78 lb/ft^3). With 2 percent water, one m^3 of ore contains 24 kg water. The 1982 assumed water demand, with 90 percent recovery required 136,360 kg of water. It will take 136,360/24 = 5682 m^3 ore. If a single 300 m-long deposit is to last a year, it must be mined to a depth of almost 15 m (49.2 ft). If the deposit is mined only to 3 m (9.84 ft) depth, 3788 m (12,425 ft) length of deposit must be mined.

C. Ore must be mined at the rate of 31.57m^3 per shift. The overburden will vary from none to 3 m^3 per m of linear advance along the deposit. As the volume of overburden varies from 3 m^3 per m of length for a mining depth of 3 m to the same volume for a mining depth of 20 m, this places limits of 3 m^3 per 1.5 m^3 of ore for each m of advance to 3 m^3 per 27 m^3 mined. Expressed another way, overburden varies from 67 percent to 10 percent of the total material mined within the mining depths listed in the table, above. Arbitrarily choosing mining depths of 5 m and 10 m, the percentages are 50 and 20 percent and the average volume of material that must be moved per shift is 52.62 m^3 at 5 m mining depth and 39.46 at 10 m. Thus a greater total volume of material must be moved at the shallower depth, to obtain the required volume of ore. It will not have to be lifted as far but if digging energy is greater than lifting energy, more total energy and time may be expended at the shallower depth.

8. Mining costs, general: Many items enter into the cost of a mining operation. Capital costs for buildings, installations and equipment are the principal ones before production begins. Labor, transportation, maintenance, processing, etc., constitute the principal post-production costs. Of the mining methods discussed in this paper, the cost of buildings and other installations should be approximately independent of mining method. A possible exception is the stationary engine scraper system in which mobile equipment is simpler but installations (towers, control booth, etc.) are more elaborate than for the other methods. The cost of both installations and equipment will be compounded of (1) original cost and (2) cost of transportation to the mining site on the moon. The latter cost is likely to be so great that the original cost probably can be ignored. Total cost of transporting each machine is arbitrarily charged to only 180 shifts (one year) in determining comparative costs per shift.

After production starts, labor is likely to dwarf all the other post-production costs listed. For example, the cost of maintenance is likely to include 80 percent or more maintenance labor. Mining crews will include an operator for each machine, one extra man and one maintenance man.

In this first order cost study it is planned to ignore installation cost differences and to base estimates solely upon (1) cost of equipment transportation from earth at $5,000 per pound of payload 18,50/ and (2) cost of lunar-based labor at $100,000 per man hour 50/. The cost of labor for an 8-hour shift thus will be $800,000 per man.

It is recognized that factors like power consumption, differences in the life of machines, manpower rotation schedules, cost of processing ore, converting to hydrogen and oxygen, storage of products and possible profits from by-products (iron, diamonds, useful fluids other than water, structural materials, sulphur, etc.) will all enter into the total economic balance between utilization of indigeneous water and the importation of water and fuels from the earth, but this study is restricted to the two major costs of surface mining alone.

9. Summary of assumptions and simplifications made for the study of lunar mining operations: Theories of lunar origin, of the origin and significance of lunar surface features, of lunar geochemistry and surface differentiation and of selenological processes and their implications, are so varied that it is necessary to impose many limitations and make many assumptions and choices, in order to arrive at any reasonable basis for the study of specific mining problems. Add to these many alternatives, a large number of proposed mining systems 1,38,39/, and it becomes even more necessary that mining conditions be narrowly defined.

Among those adopted in this study are:

(1) Mining operations will be on the side of the moon facing the earth and within 10^{o} of the lunar equator.

(2) Mining operations will be designed on such a scale as to provide one 8-hour shift operation per 24-hours or the traditional earth schedule. Thus, only one crew (shift) will be needed. Additional crews could, of course, give increased production.

(3) Mining will be done only during the lunar night or 180 shifts per (earth) year. During the daylight hours, the same crew can operate the thermal processing plant, using solar energy, if that proves the most economic energy source.

(4) Mining operations probably will be on the relatively smooth maria, these features will also be the first landing sites, the sites of the first quasi-permanent bases and the first areas to be explored.

(5) Lunar mining or excavation equipment, at first, will be much like earth-designed counterparts. One influence that may modify this will be experience gained in pre-mining lunar explorations. This influence should be a maximum on locomotion and tramming methods.

(6) Light weight metals will be used in the construction of equipment wherever it is feasible, because of the high cost of shipment to the moon. Other weight-reduction substances may also be used.

(7) Electric power, probably a-c current distributed by trailing cable, will be used for mining equipment (see Sec. V).

(8) Mobile equipment will use treads rather than wheels. Treads are heavier and slower but more rugged and universally adaptable. If wheels do prove practical, however, another weight-saving possibility will be presented.

(9) All mining systems will be compared for the year 1982 and an estimated oxygen demand of 109,100 kg/yr (240,000 lb/yr), to be met by producing 122,728 kg (270,000 lb) water per year.

(10) All ores will be assumed to have 2% water, by weight. The volume of ore (and of overburden) that must be handled will vary inversely with bulk density and a wide range is assumed for this property.

(11) It is assumed that the surface material will be scrapable (minimum depth $\sim$1 m), but the remainder of the overburden may be scrapable, may have to

be dug, or may require fragmentation prior to digging. The ore also is assumed to be scrapable, digable or must be fragmented (Section IV). It is tacitly assumed that hardness and toughness may increase with depth.

(12) For purposes of comparison of productive capacities, scraping and tramming speeds of 1.2 mph (1.93 km/hr) and of 5.2 mph (8.37 km/hr) will be assumed for scraper bulldozers, and front-end loaders. Working rate for shovels and draglines is assumed to be one cycle per 60 seconds or 480 cycles per shift.

(13) In order to be certain that production is not interrupted, it will be necessary to transport stand-by equipment to the moon. If n units are required to maintain the proper mining rate, n + 1 units will be provided up to n = 4 and n + 2 units provided when n = 5 to 10. Beyond 10 units, n + 3 units will be provided. This rule will apply to large equipment requiring only one unit, even if this unit is oversized, and may result in imposition of some extra cost burden on such units.

(14) The minimum mining crew will be three men. If only one mining machine is used this will give an operator, a maintenance man and an extra man to help with operation or maintenance. If n small units are required, n + 2 men are needed. Two maintenance men will be supplied beyond 10 <u>active</u> units, for a total of n + 3 men.

(15) Labor costs are set at $100,000 per man hour or $800,000 per man per shift giving a minimum labor cost for any operation of $\$2.40 \times 10^6$ per shift. Transport to the moon is set at $5,000 per lb of pay load, or $11,023 per kg.

IV

SURFACE MINING SYSTEMS

1. Classification of Mining Systems: After learning something specific about the occurrence of lunar minerals and certainly after some mining experience, surface mining systems may be developed which are entirely unlike any now used on earth. Until that time, however, there is little that can be done but apply variations of earth methods which seem suitable to current concepts regarding lunar mineral deposits and environment.

High unit costs and limited water demand, either for fuel or for other purposes, will require that early lunar water mining be simple, on a small scale and at a cost lower or as low as that of transportation of water from the earth. If accessible supplies of ice can be found in caverns, lava tunnels, etc., near the lunar equator, no water will be mined, in the usual sense, but it will merely be "harvested". If, however, water is found only under the lunar surface, at depths too great to be affected by the lunar diurnal temperature cycle, mining logically can be contemplated. As the demand for water, oxygen and hydrogen for lunar colonies and for fuels increases, it is reasonable to expect the size and sophistication of mining operations also to increase.

Virtually nothing has been published on "conventional" mining methods applied to lunar conditions. Heyward 37/ has discussed the general problem, pointing out the separate functions involved in mining and processing indigeneous lunar resources: rock-breaking, loading, size reduction, transportation, processing and purification, conversion and storage. Some work has appeared on indirect methods of mining, similar to the Frasch process and to solution mining 5, 38, 39/, and on tunneling methods 40/. A general review of published work on lunar mining has appeared 41/.

The work reported here deals with the more conventional surface, strip or open cast mining systems. Indirect systems 38/ and mining by drilling 1/ are also conducted from the surface but involve the removal of very limited quantities of overburden.

Systems for surface mining may be classified as follows:

(1) Scraping methods

a. Stationary engine operated scraper.

b. Scraper drawn by a tractor or other vehicle.

c. Scraping with a bulldozer.

d. Removing overburden and ore by scraper-loader.

(2) Methods involving scraping of all or part of the overburden and digging of ore or ore and part or all of overburden.

a. Overburden removed partly by scraper, bulldozer or scraper-loader and ore and part of overburden dug and loaded by front-end loader or hoe.

b. Overburden, wholly or partly removed by scraping and ore, or ore and harder overburden, dug by power shovel.

c. Same methods as b. except for ore removal by dragline.

d. Overburden and ore both dug and removed by shovel or dragline.

e. Overburden (and possibly ore) dug by ditching wheel, hoe or excavation wheel.

(3) Methods involving fragmentation of ore and of part or all of overburden prior to digging and removal.

a. Overburden removed wholly or partly by scraping and ore and part or all of overburden fragmented for removal by shovel or dragline.

b. All ore and all overburden fragmented before removal by shovel or dragline.

c. All ore and overburden fragmented but digging and loading may be done with front-end loader or hoe.

It will be noted that the methods have been roughly arranged in order of increasing hardness or toughness of overburden and ore. Surveyor I photos proved that at least part of the lunar surface is covered, to an undetermined depth, by material which probably can be scraped and removed 14 /. Under the impact hypothesis, there must be a considerable thickness of rubble both on the maria and terrae that may be scrapable if it (1) contains no or few large boulders or has not (2) become firmly cemented in some way. If the maria are younger than the terrae, the rubble blanket there probably is thinner. If the maria are volcanic tuff flows or falls, the material may be scrapable to considerable depth unless extensively intruded by lava.

The classification of mining methods is also arranged in order of increasing size of mining equipment. Small shovels and draglines will weigh more than bulldozers or loaders, with cable drawn scrapers even lighter.

Of the methods listed, only 1-a. can be made mechanically automatic or operated by one man from a single on-site location. The other methods will require an operator in a space suit or in a pressurized cab on each machine or electronic controls installed in each machine and operated from a central control tower or booth. High resolution television viewing may be required in cable-scraper or other remote control systems if overburden and ore are difficult to distinguish.

It is inferred, in the classification, that hardness or toughness of overburden and ore may increase with depth and that ore will be harder than at least a part of the overburden. These inferences appear reasonable for hydrous mineral ores, and probably, for "lean" free water ores.

2. Mining Deposit Models 1 and 2: It is obvious that, of the seven deposit models chosen, models 1 and 2 are the simplest although not necessarily the ones most likely to be encountered. It was decided to use these two deposits as the cases

by which to compare a number of mining systems. The deposits closely resemble horizontal, sedimentary earth deposits, like coal, they may be large or small and they may be mined with almost any type of excavation machinery.

No data were obtained on ditching machines and wheel excavators so these two systems will not be considered. A scraper drawn by a tractor will be somewhat different from a cable-operated scraper system and also somewhat different from a tractor-bulldozer system, but as the equipment is so near in capacity to the latter one it was decided to omit separate, detailed consideration of tractor-drawn scrapers. Previously, systems (1)c. (2)a, (2)b. and (2)c. have been compared 49 /. The same systems will be compared in a slightly different way and system (1)d. added.

A. By cable excavator system: Figure 8 shows a rope-and-scraper system for mining either deposit No. 1 or No. 2. This system is similar to larger scale tower excavators on the earth. Figure 9 is taken from a publication advertising this mining system 42 /. Such a system should be operable by one man from a central booth.

The capacity of a scraper system probably will be determined by its rate of transport rather than by its rate of digging. The transport rate is a function of scraper or bucket capacity and of the cycle frequency. Cycle frequency may vary slightly for overburden and ore. Figure 8 shows overburden deposited on a spoil pile with the ore dragged to a chute. The ore-to-chute distance is almost constant and that for waste disposal gradually increases.

One scraper manufacturer 43 / advertises heavy duty scrapers from 1.22 to 2.13 m (48 to 84 inches) wide. Weights vary from 920.8 kg (2030 lbs) to 2041.2 kg (4500 lbs) and rated capacities from 0.99 m^3 (1.29 yd^3) to 2.83 m^3 (3.70 yd^3).

Figure 8 indicates that at the stage of mining shown (another bin must be installed to the left of the one shown and the ramp extended to it), the distance of ore

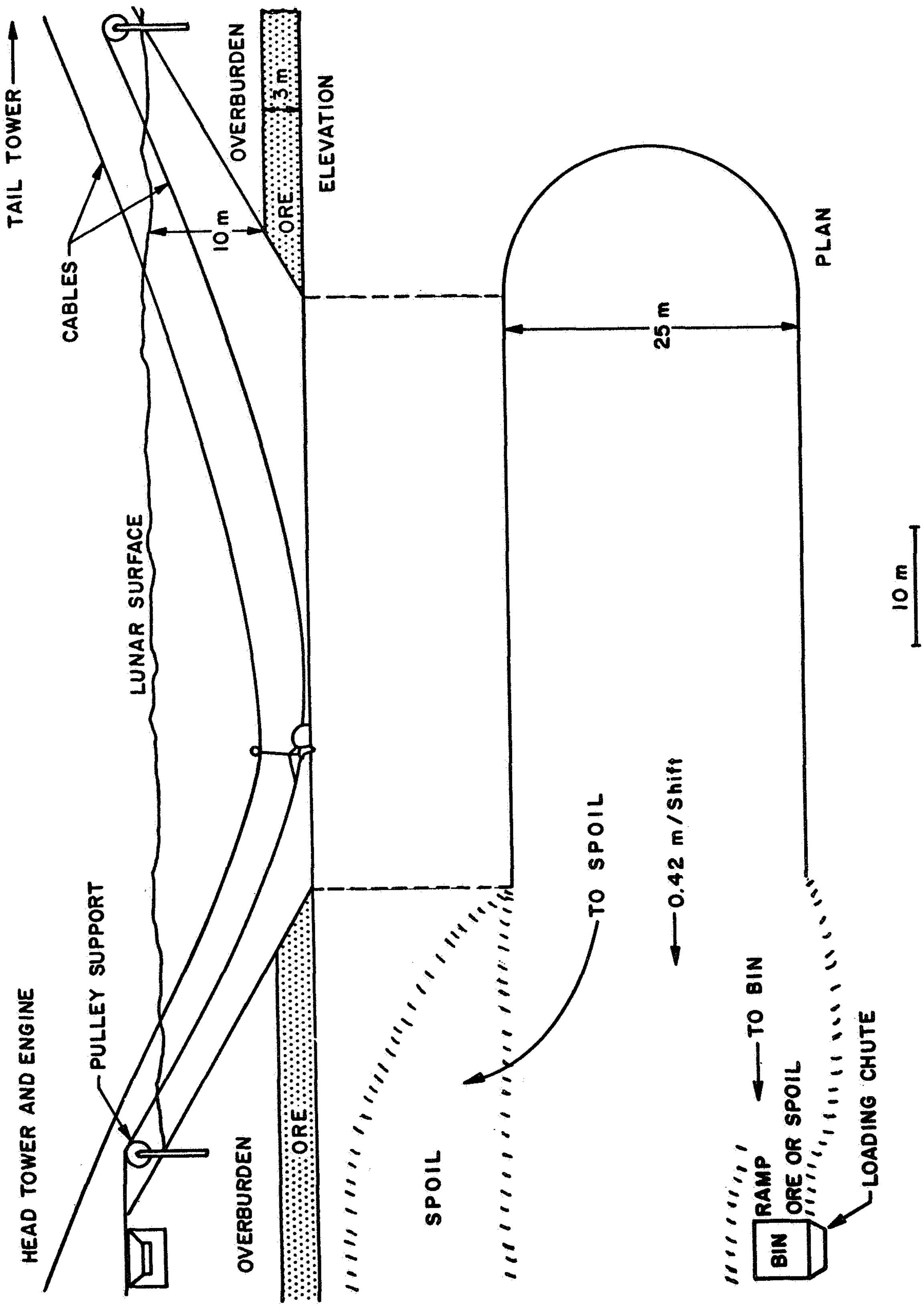


Fig. 8. Cable excavator system for mining a shallow permafrost zone.

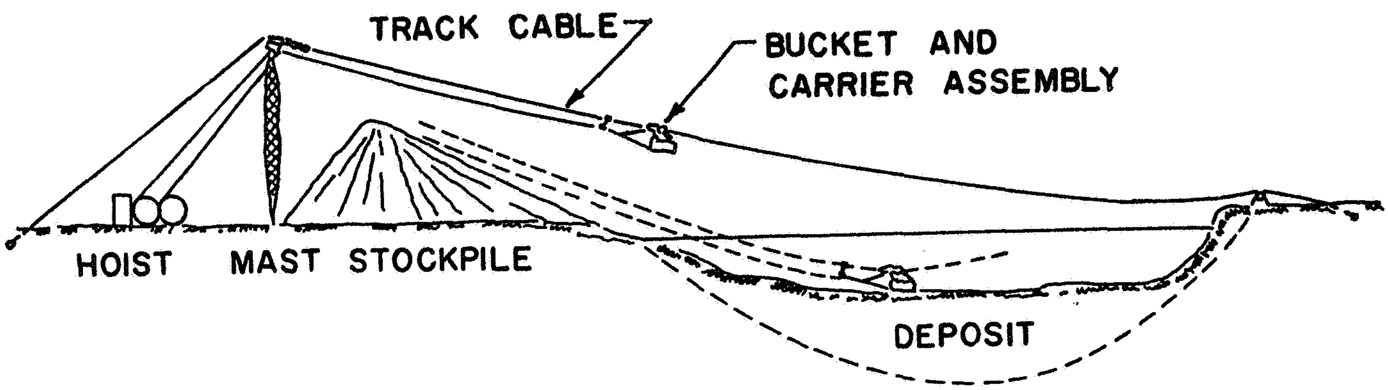


Fig. 9. Cable excavator system as illustrated by a manufacturer (Ref. 42).

transport is about 30 m. If the other bin is installed 50 meters farther left, the distance of drag will be about 80 meters. If one assumes an average drag distance for ore of 50 meters and for spoil, a longer one of 70 m, the average is about 60 m. The lowest speed listed for a small tractor was 1.2 miles per hour or 1.93 km/hr 44/. Adopting this speed as a very conservative one for dragging, recognizing that the return drag to the pit for both ore and spoil is unproductive and assuming that dumping time will require 1/2 the time required for a one-way trip, the equivalent drag distance, per productive load, is 150 meters. This gives 12.88 loads per hour or 103 loads per shift. With the 1.22 m (48-inch) scraper this is 102.0 m^3 (133.4 yd^3) and with the 2.13 m (84-inch) one, 291.5 m^3 (381.3 yd^3) per shift. Thus, the larger but not the smaller of the scrapers has sufficient capacity to secure the required output of 136.8 m^3 of material per shift, for deposit No. 1. Two of the smaller scrapers, in tandem, could mine the 10 m overburden deposit at a sufficiently rapid rate.

Deposit No. 2 requires moving of 347.2 m^3 of overburden per shift. Two of the larger scrapers, in tandem, could move that much but four of the smaller ones would be needed.

An alternative is a dragline bucket. If a 1.34 m^3 (1-3/4 yd^3) dragline bucket can be used in the place of scrapers and still attain the same dragging speeds, 138.0 m^3 per shift can be moved and the smaller deposit can be mined at the required rate.

A 3.37 m^3 (4.41 yd^3) bucket will be needed to mine the deeper overburden deposit at the same dragging speed.

If speed can be improved over that assumed, more material can be handled with a given size of scraper or dragline bucket. The highest rated speed for the small tractor having a low speed of 1.2 mph, is 5.2 mph (8.37 km/hr). Even if the small tractor cannot attain this speed under any appreciable load, it can be attained by larger and more powerful models and it might be attained by a cable scraper system. This speed will also be used as a possible "standard" one. With the same assumptions as were used for the lower speed, the faster speed should provide for 446 trips per shift. With the faster speed, the smaller 0.99 m^3 scraper can readily handle either deposit and dragline buckets as small as 0.31 m^3 (0.41 yd^3) for deposit No. 1 and 0.78 m^3 (1.02 yd^3) for No. 2 are of sufficient capacity.

One manufacturer 45 / lists heavy duty dragline buckets with the following weights and capacities:

Nominal Volume	Rated Capacity		Weight (earth)
1-3/4 yd^3	1.51 m^3 (1.97 yd^3)	and	2200 kg (4850 lbs)
1-1/4 yd^3	1.09 m^3 (1.43 yd^3)	and	1687 kg (3720 lbs)
1/2 yd^3	0.48 m^3 (0.63 yd^3)	and	862 kg (1900 lbs)

These sizes would satisfy capacity requirements for mining both deposits 1 and 2 at the fast speed and deposit 1, at the slow speed. To mine deposit No. 2 at slow dragging speed requires a 3.37 m^3 bucket or a nominal 4-1/2 yd^3 one. Another manufacturer of larger dragline buckets 46 / has a heavy duty 4-1/2 yd^3 (nominal) bucket with capacity of 3.79 m^3 (4.96 yd^3) and weighing 3493 kg (7700 lbs). This manufacturer also advertises heavy duty dragline buckets of the smaller sizes, as follows:

Nominal Volume	Rated Capacity		Weight (earth)
1-3/4 yd^3	1.47 m^3 (1.92 yd^3)	and	2132 kg (4700 lbs)
1-1/4 yd^3	1.05 m^3 (1.37 yd^3)	and	1882 kg (4150 lbs)
1/2 yd^3	0.45 m^3 (0.59 yd^3)	and	646 kg (1425 lbs)

The buckets with overlapping nominal volumes and built by the first manufacturer have a slightly higher rated capacity than those of the second one. There is no way to know if the difference is actual or due to different capacity rating systems. For calculations, the nominal 4-1/2 and 1-3/4-yd^3 buckets of the second named manufacturer will be used and the 1-1/4- and 1/2-yd^3 buckets of the first one.

Looking at the mechanics of operating a rope-and-scraper system with a forward or (engine) end and a tail pulley, it can be seen that a rigid-position set-up will make it almost impossible to run a wide pit. The scraper will be difficult to direct to all parts of an essentially rectangular pit without frequent moves of head or tail tower or without use of side ropes and pulleys to guide the scraper, first to the spoil pile and than to the ore bin. A round pit, with the head end in the center, would require frequent moving of only the tail pulley. A narrow pit would require frequent moves forward of both towers. Undoubtedly, this method cannot be quite as productive as has been calculated because some allowance must be made for the time required for moves.

To estimate the weights of cable excavator systems the following approximations were made:

(1) a 15.2 m (50 ft) high head tower and a 7.6 m (25 ft) tail tower are assumed for deposit No. 1 with a 30.5 m (100 ft) head tower for deposit No. 2. Assuming 9"x9" steel channels, 1/2" thick, for the 4 tower supports, a total length of 125.0 m (410 ft) will be needed for the principal support members (they will not stand vertically) for the tallest tower, 62.5 m (205 ft), for the lower head tower, and 31.4 m (103 ft) for the tail tower. Cross members will be lighter but of a much greater linear total, so it is assumed the entire weight of cross members will equal that of the 4 principal supports. This will give 33,286 kg (73,382 lbs) for the weight of the taller head tower; 16,652 kg (36,711 lbs) for the shorter one and 8,366 kg (18,444 lbs) for the tail tower, if all are made of steel. All or part of the steel may be replaced by Ti-6Al-4V or other strong light metal alloy but to be on the conservative side, only 1/2 the secondary members or bracing will be assumed to be made of Ti-6Al-4V (density = 4.43 gm/cm^3). This replacement will give earth weights of:

30.5 m head tower	26,035 kg (57,397 lbs)
15.2 m " "	13,025 kg (28,711 lbs)
7.6 m tail "	6,544 kg (14,427 lbs)
	45,604 kg (100,535 lbs)

All three towers will be shipped for deposit No. 2. They will be designed so that bracing and support sections are interchangeable. One head and the tail tower will be assembled and the other kept for parts to be assembled as a substitute head or tail tower. Deposit No. 1 will require only two of the shorter head towers and the tail tower.

(2) A maximum of 915 m (3000 ft) of 1/2" stainless steel cable will be needed but double this quantity should be shipped so that the equivalent of a spare cable will be available. One manufacturer 47/ provides a 16x19 IWRC cable with a breaking

strength of 10,364 kg (22,800 lbs) and a weight per 30.5 m (100 ft) of 26.8 kg (59.0 lbs). Without spools, 1830 m of cable will weigh 1609 kg (3540 lbs). A total weight of 1700 kg (3748 lbs) was assumed to include a spool or spools for convenience and compactness in shipping and for convenient handling. Cables made of light-weight alloys may possibly be substituted here but as the total weight is low, this will not be done.

(3) It is estimated that the maximum sized motor required for deposit No. 1 or No. 2 should be about 149.2 KW (200 HP). At 3.0 kg/kwhr (see Sec V) this will give a weight of 447.6 kg (987 lbs). Two motors will be provided in the case of failure giving a weight of 900 kg (1984 lbs).

The total weight of the system thus is 48,204 kg (106,271 lbs) plus two of the scrapers or dragline buckets selected for the deposit. Deposit No. 1 will require only a weight of 33,384 kg. Table 1 shows the essential physical data and costs for both deposits No. 1 and 2 and for both scrapers and the four sizes of dragline buckets.

Table 1 indicates practically identical costs, per shift, for dragline buckets, mining deposit No. 1 at the slow speed. Of course, deposit No. 2 can be mined only with the 4-1/2 yd^3 dragline bucket, at this speed, although this unit is least economical for deposit No. 1.

At the higher dragging speed, the 1.22 m scraper and the 1/2 yd^3 bucket are most economical for deposit No. 1 and the 1.22 m scraper, for deposit No. 2.

It is to be expected that equipment failures, like a snapped cable, broken bucket teeth, dull scraper blade, etc., will occur more frequently, in time, with the higher speed although the incidence of such events, per m^3 of material moved, may not be greater. The frequency with which large boulders or large masses of meteoritic iron, requiring special handling, will occur, may determine preferable dragging speed although general toughness of the material with regard to scraping, probably will be more determinative.

Table 1

Capacity, required number of units and personnel and first order costs of Cable Excavator Systems for mining deposits No. 1 and No. 2

Machine No.	Machine type or size	Machine Capacity m^3	Machine Capacity yd^3	Capacity/Shift Deposit m^3	Capacity/Shift Deposit yd^3	Units Needed (2) No. 1	Units Needed (2) No. 2	System Wt., Deposit No. 1 Kg	System Wt., Deposit No. 1 lb.	System Wt., Deposit No. 2 No. 1	System Wt., Deposit No. 2 No. 2	Crew Size No. 1	Crew Size No. 2	Transport Cost, \$ No. 1	Transport Cost, \$ No. 2	Personnel Cost, \$ No. 1	Personnel Cost, \$ No. 2	Total Cost No. 1	Total Cost No. 2
At 1.93 Km/hr. dragging speed or 103 trips per shift																			
1	1.22 m (1)	0.99	1.29	102.0	133.4	2	4	- (3)	-	-	-	-	-	-	-	-	-	-	-
2	2.13 m (1)	2.83	3.70	291.5	381.3	1	2	37,500	82,700	-	-	3	-	$2.30x10^6$	-	$2.4x10^6$	-	$4.70x10^6$	-
3	0.50 yd^3 (4)	0.48	0.63	49.4	64.9	3	8	-	-	-	-	-	-	-	-	-	-	-	-
4	1.25 yd^3 (4)	1.09	1.43	112.3	147.3	2	4	-	-	-	-	-	-	-	-	-	-	-	-
5	1.75 yd^3 (4)	1.47	1.92	151.4	197.8	1	3	37,700	83,100	-	-	3	-	$2.31x10^6$	-	$2.4x10^6$	-	$4.71x10^6$	-
6	4.50 yd^3 (4)	3.79	4.96	390.4	510.9	1	1	40,390	89,100	55,200	121,700	3	3	$2.48x10^6$	$3.38x10^6$	$2.4x10^6$	$2.4x10^6$	$4.88x10^6$	$5.78x10^6$
At 8.37 Km/hr dragging speed or 446 trips per shift																			
1	1.22 m (1)	0.99	1.29	441.5	575.3	1	1	35,300	77,800	50,100	110,500	3	3	$2.16x10^6$	$3.07x10^6$	$2.4x10^6$	$2.4x10^6$	$4.56x10^6$	$5.47x10^6$
2	2.13 m (1)	2.83	3.70	1262.2	1650.2	1	1	37,500	82,700	52,300	115,300	3	3	$2.30x10^6$	$3.20x10^6$	"	"	$4.70x10^6$	$5.60x10^6$
3	0.50 yd^3 (4)	0.48	0.63	214.1	281.0	1	2	35,100	77,400	-	-	3	-	$2.15x10^6$	-	"	"	$4.55x10^6$	-
4	1.25 yd^3 (4)	1.09	1.43	486.1	637.8	1	1	36,800	81,200	51,600	113,800	3	3	$2.26x10^6$	$3.16x10^6$	"	"	$4.66x10^6$	$5.56x10^6$
5	1.75 yd^3 (4)	1.47	1.92	655.6	856.3	1	1	37,700	83,100	52,500	115,800	3	3	$2.31x10^6$	$3.22x10^6$	"	"	$4.71x10^6$	$5.62x10^6$
6	4.50 yd^3 (4)	3.79	4.96	1690.3	2212.2	1	1	40,390	89,100	55,200	121,700	3	3	$2.48x10^6$	$3.38x10^6$	"	"	$4.88x10^6$	$5.78x10^6$

(1) Scrapers. Dimension given is width.
(2) This number includes no provision for stand-by units.
(3) Blanks are left in all columns calling for more than one scraper or dragline bucket because it is doubtful if the use of two or more units in tandem on the same cable will prove feasible.
(4) Nominal capacity of dragline bucket.

B. By tractor-drawn scrapers: An alternative, and much more flexible scraping system, would be a scraper drawn by a tractor. This method probably will require an operator in a space suit or in a tractor cab. It should also prove an ideal method, if it becomes necessary to mine a large number of small, scattered deposits rather than a large one.

The pit layout could be similar to that shown in Fig. 8, but with much more latitude as to location of spoil piles and loading chutes.

The limiting factors to material transportation capacity for this mining method probably will be very close to that of the previous one. It is possible that the average length of haul could be shortened. The haulage speed probably will be much nearer 5.2 than 1.93 kmphr but scraping speed, even for larger tractors, probably will be nearer the lower speed. No calculations were made for this method. Equipment weights will be quite similar to those estimated in the next method to be discussed.

C. By tractor-bulldozers: A very similar mining method to the two preceding ones is a method in which ore and overburden are scraped by a bulldozer to loading chute or spoil pile. Material is dug by the blade and then pushed to the disposal point. This method is easily as flexible and as adaptable to small deposits as is the one discussed just above.

A very wide variety of bulldozers are available. The limiting capacity factor again will be, not digging rate, but transporting rate. Using the assumption of material piled against the blade at a 45° angle, Table 2 shows the capacities of seven small dozers on which information was collected. It also shows the capacities of each at 1.93 km/hr (1.2 mph) and at 8.37 km/hr (5.2 mph). The minimum number of bulldozers and the crew size required to handle both 10 and 30 meters of overburden, plus 3 m of ore, at the both speeds, are also shown.

Considerably larger bulldozers are made than those listed in Table 2, but they are also much heavier. A mining installation using this system will need a minimum

Table 2

Blade dimensions, capacity, weight, and transport, personnel and total first-order cost of seven bulldozers (all crawlers).

Machine Number	Blade Size, in.	Transport Capacity		Capacity Per Shift		No. Machines Needed (1)		System Wt. Dep. No. 2 (4)		System Wt. Dep. No. 2 (4)		Crew Size		Transport Cost, \$		Personnel Cost, \$		Total Cost, \$ Per Shift	
		m^3	yd^3	m^3	yd^3	No. 1	No. 2	kg	lbs	kg	lbs	No. 1	No. 2	No. 1	No. 2	No. 1	No. 2	No. 1	No. 2
At 1.93 km/hr tractor speed or 103 trips per shift																			
1	80x24	0.38	0.49 (2)	38.9	50.9	5	11	14,140	31,200	31,110	68,600	6	12	0.89×10^6	1.91×10^6	4.80×10^6	9.60×10^6	5.67×10^6	11.51×10^6
2	80x24	0.38	0.49 (2)	38.9	50.9	5	11	16,510	36,400	36,310	80,100	6	12	1.01×10^6	2.23×10^6	4.80×10^6	9.60×10^6	5.81×10^6	11.83×10^6
3	83.9x28	0.54	0.71 (2)	55.5	72.6	4	9	15,810	34,900	35,580	78,400	5	9	0.97×10^6	2.18×10^6	4.00×10^6	7.20×10^6	4.97×10^6	9.38×10^6
4	110x28	0.71	0.92 (3)	72.8	95.2	3	7	14,990	33,000	34,990	77,100	4	7	0.92×10^6	2.14×10^6	3.20×10^6	5.60×10^6	4.12×10^6	7.75×10^6
5	122x28	0.74	0.97 (3)	76.2	99.7	3	7	15,440	34,000	36,040	79,500	4	7	0.94×10^6	2.21×10^6	3.20×10^6	5.60×10^6	4.14×10^6	7.81×10^6
6	96x33.2	0.87	1.13 (3)	89.3	116.8	3	5	15,440	34,000	25,740	56,800	4	6	0.94×10^6	1.58×10^6	3.20×10^6	4.80×10^6	4.14×10^6	6.38×10^6
7	125x34	1.18	1.55 (3)	121.8	159.2	3	4	21,160	46,600	28,220	62,200	4	5	1.29×10^6	1.73×10^6	3.20×10^6	4.00×10^6	4.49×10^6	5.75×10^6
At 8.37 km/hr tractor speed or 446 trips per shift																			
1	80x24	0.38	0.49 (2)	169.5	221.7	2	4	5,660	12,480	11,310	24,930	3	5	0.35×10^6	0.69×10^6	2.40×10^6	4.00×10^6	2.75×10^6	4.69×10^6
2	80x24	0.38	0.49 (2)	169.5	221.7	2	4	6,600	14,550	13,200	29,100	3	5	0.40×10^6	0.81×10^6	2.40×10^6	4.00×10^6	2.80×10^6	4.81×10^6
3	83.9x28	0.54	0.71 (2)	240.8	374.9	2	3	7,910	17,440	11,860	26,100	3	4	0.48×10^6	0.73×10^6	2.40×10^6	3.20×10^6	2.88×10^6	3.93×10^6
4	110x28	0.71	0.92 (3)	316.7	414.2	2	3	10,000	22,050	10,000	22,050	3	4	0.61×10^6	0.61×10^6	2.40×10^6	3.20×10^6	3.01×10^6	3.81×10^6
5	122x28	0.74	0.97 (3)	330.0	431.6	2	3	10,300	22,710	10,300	22,710	3	4	0.63×10^6	0.63×10^6	2.40×10^6	3.20×10^6	3.03×10^6	3.83×10^6
6	96x33.2	0.87	1.13 (3)	388.0	507.5	2	2	10,300	22,710	10,300	22,710	3	4	0.63×10^6	0.63×10^6	2.40×10^6	3.20×10^6	3.03×10^6	3.83×10^6
7	125x34	1.18	1.55 (3)	526.3	688.3	2	2	14,110	31,110	14,110	31,110	3	3	0.86×10^6	0.86×10^6	2.40×10^6	2.40×10^6	3.26×10^6	3.26×10^6

(1) This <u>includes</u> the assumed number of stand-by machines. See Sec. III 8.
(2) Gasoline or diesel driven models.
(3) Diesel engines only.
(4) Based on 72% of earth model weight.

of two machines in order to keep production going in case maintenance is needed on the principal mining machine.

Weights in Table 2 were calculated from those supplied by manufacturers by the formula: 0.12 (non-steel material part of it lighter than steel) + 0.88 (Ti-6Al-4V, and a little Al replacing steel) = 0.72 or the lunar design will weigh 72% of the commercial model. Further savings may be made in motors or by special weight-saving design (section II4. and V) but the first order estimate will use only the 0.72 factor.

Table 2 shows bulldozer No. 4 to give the lowest cost for mining deposit No. 1. It is followed so closely by bulldozers No. 5 and 6 that it is fair to say that these three models seem to represent the optimum size and capacity for this deposit. The heaviest bulldozer was also close but the three lightest models were considerably less economic. The small ones had lower system weights but higher personnel requirements.

In the case of deposit No. 2, the largest bulldozer was the most economic and the next largest, was second most economic. It is possible that a model large enough to mine the deposit with one unit and a minimum crew might prove even more economical because personnel cost ($\$3.20 \times 10^6$ minimum) is the predominant cost in every case.

D. <u>By front-end loaders</u>: A pit layout similar to that of Fig. 8 may be preserved if overburden and ore are dug, transported and dumped by front-end loaders. Table 3 gives data for 12 small front-end loaders. The bucket capacity (level rather than heaped, where both are given) is the average of the range given by the manufacturer. It is evident that for equivalent capacity, rubber tired models are considerably lighter than crawlers. For consistency, however, only the crawler models should be used for final cost comparisons, although costs are calculated for 6 wheeled models. If wheeled vehicles prove satisfactory for haulage on the lunar surface, there exist additional possibilities for weight reductions. Even if rubber

Table 3

Table showing capacities, weights, personnel requirements and transport, personnel and total costs for 6 wheeled and 6 crawler front-end loaders.

Machine	Traction Mode	Bucket Capacity M^3	Bucket Capacity Yd^3	Capacity Per Shift M^3	Capacity Per Shift Yd^3	No. Machines Needed No. 1	No. Machines Needed No. 2	System Wt. Dep. No. 1 Kg	System Wt. Dep. No. 1 Lbs.	System Wt. Dep. No. 2 Kg	System Wt. Dep. No. 2 Lbs.	Crew Size No. 1	Crew Size No. 2	Transport Cost, \$ Per Shift No. 1	Transport Cost, \$ Per Shift No. 2	Personnel Cost, \$ Per Shift No. 1	Personnel Cost, \$ Per Shift No. 2	Total Cost, \$ Per Shift No. 1	Total Cost, \$ Per Shift No. 2
At 103 trips per shift																			
1(3)	wh	0.99	1.30	102.4	133.9	3	5	5,060	11,160	8,420	18,570	4	6	0.31×10^6	0.52×10^6	3.20×10^6	4.80×10^6	3.51×10^6	5.32×10^6
2(2)	wh	1.06	1.38	108.6	142.1	3	5	9,460	20,860	15,770	34,770	4	6	0.58×10^6	0.97×10^6	3.20×10^6	4.80×10^6	3.78×10^6	5.77×10^6
3(3)	Cr	0.62	0.81	63.8	83.4	4	8	13,730	30,270	27,460	60,940	5	8	0.84×10^6	1.69×10^6	4.00×10^6	6.40×10^6	4.84×10^6	8.09×10^6
4(3)	wh	1.25	1.63	128.4	167.9	3	4	10,400	22,930	13,860	30,560	4	5	0.64×10^6	0.85×10^6	3.20×10^6	4.00×10^6	3.84×10^6	4.85×10^6
5(3)	Cr	0.76	1.00	78.8	103.0	3	7	12,260	27,030	28,610	63,080	4	7	0.75×10^6	1.75×10^6	3.20×10^6	5.60×10^6	3.95×10^6	7.35×10^6
6(4)	Cr	0.86	1.13	89.0	116.4	3	5	14,540	32,060	24,230	53,420	4	6	0.89×10^6	1.48×10^6	3.20×10^6	4.80×10^6	4.09×10^6	6.28×10^6
7(4)	wh	1.53	2.00	157.5	206.0	2	4	10,920	24,080	21,840	48,150	3	5	0.67×10^6	1.34×10^6	2.40×10^6	4.00×10^6	3.07×10^6	5.34×10^6
8(4)	wh	1.72	2.25	177.2	231.8	2	3	11,070	24,410	16,600	36,600	3	4	0.68×10^6	1.02×10^6	2.40×10^6	3.20×10^6	3.08×10^6	4.22×10^6
9(4)	Cr	1.00	1.31	103.1	134.9	3	5	17,160	37,830	28,600	63,060	4	6	1.05×10^6	1.75×10^6	3.20×10^6	4.80×10^6	4.25×10^6	6.55×10^6
10(4)	wh	1.53	2.00	157.5	206.0	2	4	12,800	28,220	25,590	56,420	3	5	0.78×10^6	1.57×10^6	2.40×10^6	4.00×10^6	3.18×10^6	5.57×10^6
11(4)	Cr	1.15	1.50	118.1	154.5	3	4	20,370	44.910	27,160	59,880	4	5	1.25×10^6	1.66×10^6	3.20×10^6	4.00×10^6	4.45×10^6	5.66×10^6
12(4)	Cr	1.15	1.50	118.1	154.5	3	4	21,090	46,500	28,120	62,000	4	5	1.29×10^6	1.72×10^6	3.20×10^6	4.00×10^6	4.49×10^6	5.72×10^6
At 446 trips per shift																			
1	wh	0.99	1.30	441.5	577.5	2	2	3,370	6,830	3,370	6,830	3	3	0.19×10^6	0.19×10^6	2.40×10^6	2.40×10^6	2.59×10^6	2.59×10^6
2	wh	1.06	1.38	472.8	618.4	2	2	6,310	12,780	6,310	12,780	3	3	0.36×10^6	0.36×10^6	2.40×10^6	2.40×10^6	2.76×10^6	2.76×10^6
3	Cr	0.62	0.81	276.5	361.6	2	3	6,870	13,910	10,300	20,850	3	4	0.39×10^6	0.58×10^6	2.40×10^6	3.20×10^6	2.79×10^6	2.98×10^6
4	wh	1.25	1.63	557.5	729.2	2	2	6,930	14,030	6,930	14,030	3	3	0.39×10^6	0.39×10^6	2.40×10^6	2.40×10^6	2.79×10^6	2.79×10^6
5	Cr	0.76	1.00	339.0	443.4	2	3	8,180	16,560	12,260	24,820	3	4	0.46×10^6	0.69×10^6	2.40×10^6	3.20×10^6	2.86×10^6	3.09×10^6
6	Cr	0.86	1.13	383.6	501.7	2	2	9,700	19,640	9,700	19,640	3	3	0.55×10^6	0.55×10^6	2.40×10^6	2.40×10^6	2.95×10^6	2.95×10^6
7	wh	1.53	2.00	682.4	892.5	2	2	10,920	22,110	10,920	22,110	3	3	0.61×10^6	0.61×10^6	2.40×10^6	2.40×10^6	3.01×10^6	3.01×10^6
8	wh	1.72	2.25	767.1	1003.3	2	2	11,070	22,410	11,070	22,410	3	3	0.62×10^6	0.62×10^6	2.40×10^6	2.40×10^6	3.02×10^6	3.02×10^6
9	Cr	1.00	1.31	446.0	583.3	2	2	11,440	23,160	11,440	23,160	3	3	0.64×10^6	0.64×10^6	2.40×10^6	2.40×10^6	3.04×10^6	3.04×10^6
10	wh	1.53	2.00	682.4	892.5	2	2	12,800	25,910	12,800	25,910	3	3	0.72×10^6	0.72×10^6	2.40×10^6	2.40×10^6	3.12×10^6	3.12×10^6
11	Cr	1.15	1.50	512.9	670.8	2	2	13,580	27,490	13,580	27,490	3	3	0.76×10^6	0.76×10^6	2.40×10^6	2.40×10^6	3.16×10^6	3.16×10^6
12	Cr	1.15	1.50	512.9	670.8	2	2	14,060	28,470	14,060	28,470	3	3	0.79×10^6	0.79×10^6	2.40×10^6	2.40×10^6	3.19×10^6	3.19×10^6

(1) Wh - wheeled (rubber-tired) model
Cr - crawler model with steel treads
(2) Gasoline motor
(3) Gasoline or diesel
(4) Diesel only

tires prove impractical, large, light metal wheels may prove sufficiently strong and lighter than crawlers.

The percentage of steel in front-end loaders is almost identical to that in tractors, based on the small quantity of data obtained from manufacturers. It was assumed that light metal substitution for steel will be the same in the two and the weight correction factor of 0.72 was used for front-end loader models also.

Front-end loaders dig rather than scrape, then transport and dump. For comparison, the 103 and 446 trips per shift used for bulldozers, will be used as cycles per shift for front-end loaders. This assumption may or may not be valid and without verification, introduces a possible discontinuity in any method or scale of comparison of equipment capacities and costs.

Table 3 shows the same preponderance of personnel costs over transport costs as was shown for bulldozers. Weight does not appear to be proportional to bucket capacity, even when crawler models only are considered. The result probably is due to differences in the type of work for which the loader was designed. Loading broken rock, including some large blocks, requires more rugged construction and hence more weight, than loading gravel or soil.

Among crawler models, front-end loader No. 5 is most economical for deposit No. 1 and No. 11 for No. 2. Deposit No. 1 requires so few machines that low weight per unit capacity was a great advantage. For the larger deposit, however, high capacity per machine is most important. Crawler models 11 and 12 have the same capacity but No. 12 is second because it weighs a little more than 11. The other 4 models all have lower capacities than Nos. 11 and 12. No. 3, with the lowest capacity, is sixth because, among crawlers, system weight is not only third highest but 6 machines must be operated and two stand-bys provided.

Among wheeled models, the highest capacity machine, No. 8, ranks first for the large deposit and second for the small one because the minimum number of working machines are required. Wheeled model No. 8 is considerably oversized for deposit No. 1 and it loses first place there to No. 7 which is only slightly oversized and is lighter. Fitting the capacity and weight of the machine to required mining rate thus seems to be an important factor.

E. By scraper-loaders: The scraper-loader, often referred to simply as a scraper or carryall 48 /, refers to a machine that scrapes earth, loading as it goes and then hauls the load to its destination where it is dumped. Some large models have material elevators for heaped loading but this type was not considered.

Only three scraper-loaders were considered. Two of them weigh practically the same and have the same capacity. Each of the two is the smallest model advertised by its manufacturer. Both have integral motors and rubber tired wheels. The third model is much larger. No crawlers were advertised by either manufacturer. There are towed models 48 / which may be towed by a crawler tractor. Such a combination with like capacity, may be some heavier than the models considered here.

Because the rather large body probably can be made of light metal alloy a weight modification formula of 25% steel or equivalent and 75% Ti-6Al-4V, giving a factor of 0.67, is used. Table 4 shows the capacity and cost calculation results. When a scraper-loader is scraping it will be in a low gear and traveling slowly; when hauling to destination, it should travel faster. For consistency, if not accuracy, the equivalent average speed used with other traveling units to give 103 and 446 loads per 8-hour shift, are used here. Reference to Table 4 will show that the equipment is so greatly oversized for both deposits, at 103 loads per shift, that calculations at 446 loads would be meaningless and they are omitted.

With equipment sized so that only one active and one stand-by unit are needed, all crews are minimum and transport weight alone will determine relative economy.

Table 4

Table showing capacities, weights and transport, personnel and total costs for three models of scraper-loaders

Machine Number	Machine Capacity		Capacity Per Shift (1)		No. Machines (2) Needed, Deposit		System Wt. Deposit No.1		System Wt. Deposit No.2		Crew Size		Transport Costs, $/Shift		Personnel Costs, $/Shift		Total Cost $ Per Shift	
	m^3	yd^3	m^3	yd^3	No. 1	No. 2	kg	lbs	kg	lbs	No.1	No. 2	No. 1	No. 2	No. 1	No. 2	No. 1	No. 2
1	6.12	8.00	630.4	824.0	2	2	16,110	35,510	16,110	35,510	3	3	0.99×10^6	0.99×10^6	2.40×10^6	2.40×10^6	3.39×10^6	3.39×10^6
2	6.12	8.00	630.4	824.0	2	2	15,930	35,110	15,930	35,110	3	3	0.92×10^6	0.92×10^6	2.40×10^6	2.40×10^6	3.32×10^6	3.32×10^6
3	10.70	14.00	1,102.1	1,442.0	2	2	25,650	56,550	25,650	56,550	3	3	1.57×10^6	1.57×10^6	2.40×10^6	2.40×10^6	3.97×10^6	3.97×10^6

(1) At 103 trips/shift

(2) Includes stand-by machine

If transportation of ore to the mill should be done by the equipment that mines it, scraper-loaders will be ideal. Their use will, however, be unlikely if large boulders are present or if the lunar material is solid and tougher than soil or softer clays on earth.

F. By shovels and draglines: Both the deposits can also be mined by shovels and draglines. Because the "reach" of both types of machines is limited and their mobility restricted, some kind of ore haulage will be needed. To place the ore mined by shovels in bins located as they were in the previous examples would, however, require only very short hauls. If it is assumed that ore, regardless of the system of mining, will be hauled to the preparation plant, bins are unnecessary for any of the nonscraping methods, including shovels and draglines, because the ore may be loaded directly into trucks or simply be piled on the surface for later loading and hauling. If ore must be hauled to the processing plant an appreciable distance, systems using front-end loaders, shovels and draglines will not need to be charged with any greater haulage costs than will scraper methods.

Draglines have greater capability for mining below machine level than do shovels although the latter can make "box" or first cuts, after which all mining can be at machine level and above, in a deposit like that shown in Fig. 8. The shovel generally will work in the pit bottom while the dragline works from the bank. With 10 m thick overburden, all the machines shown in Table 5 should be able to mine all material in one bench. For 30 m overburden depth, removal in not less than two benches, will be necessary.

Loads that can be picked up by draglines decrease as boom length becomes greater. On the moon, both boom weight and equivalent-volume load are less than on earth so that boom lengths and consequently, dragline mining depths, may be correspondingly greater. The same conclusions with regard to boom lengths and mining heights should be true of shovels, but to a lesser degree.

Table 5

Table showing power, capacity, range, weights and transport personnel, and total costs for 5 power shovels and 9 draglines.

Machine Number	Power		Bucket Capacity(1)		Oper. Radius or Depth(2)		Capacity Per Shift		System Wt. Dep. No. 1		System Wt. Dep. No. 2		Crew Size		Transport Cost, $/Shift		Personnel Cost, $/Shift		Total Cost $ Per Shift	
	kw	hp	m^3	yd^3	m	ft	m^3	yd^3	kg	lbs	kg	lbs	No. 1	No. 2	No. 1	No. 2	No. 1	No. 2	No. 1	No. 2
SHOVELS																				
1	112	150	1.44	1.88	9.0	29.5	691	904	62,230	137,190	62,230	137,190	3	3	3.81×10^6	3.81×10^6	2.40×10^6	2.40×10^6	6.21×10^6	6.21×10^6
2	153	205	2.29	3.00	12.1	39.8	1,099	1,437	93,350	205,800	93,350	205,800	3	3	5.72×10^6	5.72×10^6	2.40×10^6	2.40×10^6	8.12×10^6	8.12×10^6
3	183	245	2.68	3.50	11.5	37.6	1,286	1,682	114,490	252,400	114,490	252,400	3	3	7.01×10^6	7.01×10^6	2.40×10^6	2.40×10^6	9.41×10^6	9.41×10^6
4	112	150(3)	1.91	2.50	9.5	31.0	917	1,199	99,280	218,860	99,280	218,860	3	3	6.08×10^6	6.08×10^6	2.40×10^6	2.40×10^6	8.48×10^6	8.48×10^6
5	172	230	1.91	2.50	11.3	32.0	917	1,199	113,900	251,100	113,900	251,100	3	3	6.98×10^6	6.98×10^6	2.40×10^6	2.40×10^6	9.38×10^6	9.38×10^6
DRAGLINES																				
1	112	150	1.44	1.88	13.7	45.0	691	904	67,480	148,760	67,480	148,760	3	3	4.13×10^6	4.13×10^6	2.40×10^6	2.40×10^6	6.53×10^6	6.53×10^6
2	153	205	2.48	3.75	16.8	55.0	1,190	1,556	83,620	184,340	83,620	184,340	3	3	5.12×10^6	5.12×10^6	2.40×10^6	2.40×10^6	7.52×10^6	7.52×10^6
3	183	245	2.68	3.50	16.8	55.0	1,286	1,682	96,410	212,550	96,410	212,550	3	3	5.90×10^6	5.90×10^6	2.40×10^6	2.40×10^6	8.30×10^6	8.30×10^6
4	172	230	1.72	2.25	----	-------	826	1,080	101,240	223,200	101,240	223,200	3	3	6.20×10^6	6.20×10^6	2.40×10^6	2.40×10^6	8.60×10^6	8.60×10^6
5	112	150	1.82	2.38	----	-------	874	1,143	104,620	230,640	104,620	230,640	3	3	6.41×10^6	6.41×10^6	2.40×10^6	2.40×10^6	8.81×10^6	8.81×10^6
6	195	262	2.01	2.63	----	-------	965	1,262	108,560	239,320	108,560	239,320	3	3	6.65×10^6	6.65×10^6	2.40×10^6	2.40×10^6	9.05×10^6	9.05×10^6
7	112	150(3)	2.20	2.88	----	-------	1,056	1,381	117,850	259,800	117,850	259,800	3	3	7.22×10^6	7.22×10^6	2.40×10^6	2.40×10^6	9.62×10^6	9.62×10^6
8	92	123	1.15(4)	1.50	10.7	35.0(4)	552	722	51,930	114,480	51,930	114,480	3	3	3.19×10^6	3.19×10^6	2.40×10^6	2.40×10^6	5.59×10^6	5.59×10^6
9	224	300	2.68(4)	3.50	12.2	40.0(4)	1,286	1,682	116,830	257,560	116,830	257,560	3	3	7.15×10^6	7.15×10^6	2.40×10^6	2.40×10^6	9.55×10^6	9.55×10^6

(1) Average of the range given by the manufacturer
(2) Operating radius for shovels and digging depth for draglines with longest boom where ascertainable from manufacturers literature
(3) Electric power. Figure is for largest motor. Three smaller motors present. Estimated max. power at any one time, 186 kw (250 hp).
(4) Not given but estimated from similar sized equipment

A shovel, swinging the boom to dumping position and dumping, is different operation from that of scraping. The dragline can be said to scrape during part of its cycle but during the other part, the action is more like that of the shovel in the latter part of its cycle.

Observation of a skilled operator will give shovel or dragline normal working cycle time but many factors lengthen effective cycle time. The principal one of these is time for moving the shovel or dragline along the face.

For shovels of 3/8 yd^3 to 3-1/2 yd^3, nominal bucket size, Nichols 48/ gives cycle times of 13 to 22 seconds as common on earth. This, of course, allows no time for moving the machine, etc. Fig. 10 shows the relation of volume of material handled, per 8-hour shift, to bucket size, for cycle times of 30, 45, 60, 75 and 90 seconds or for 960, 640, 480, 384 and 320 cycles in an 8-hour shift. All cycles are longer than the longest one cited by Nichols 48/, thus allowing some time for machine moving. As shovels and draglines probably would be used only on comparatively larger lunar deposits where machine moves would not be frequent, 30-second cycles may be a liberal average. In order to allow more conservative estimates, however, all calculations for Table 5 have been made with the 60-second cycle or 480 capacity loads per 8-hour shift.

After becoming accustomed to the lower lunar gravity, an operator should be able to handle a shovel or dragline as well as on earth. During the swing to the dumping position, a dragline bucket is under less control than a shovel bucket. It is suspended and is moving laterally at the speed of swing and upward, to dumping height, against gravity. This means that gravity is operating at 90° to the swing force. With the lower lunar gravity, for a given swing velocity, the gravity component is less and the angle of the bucket line with the horizontal should be smaller than on earth. On stopping at the dump point, this condition and the lack of any

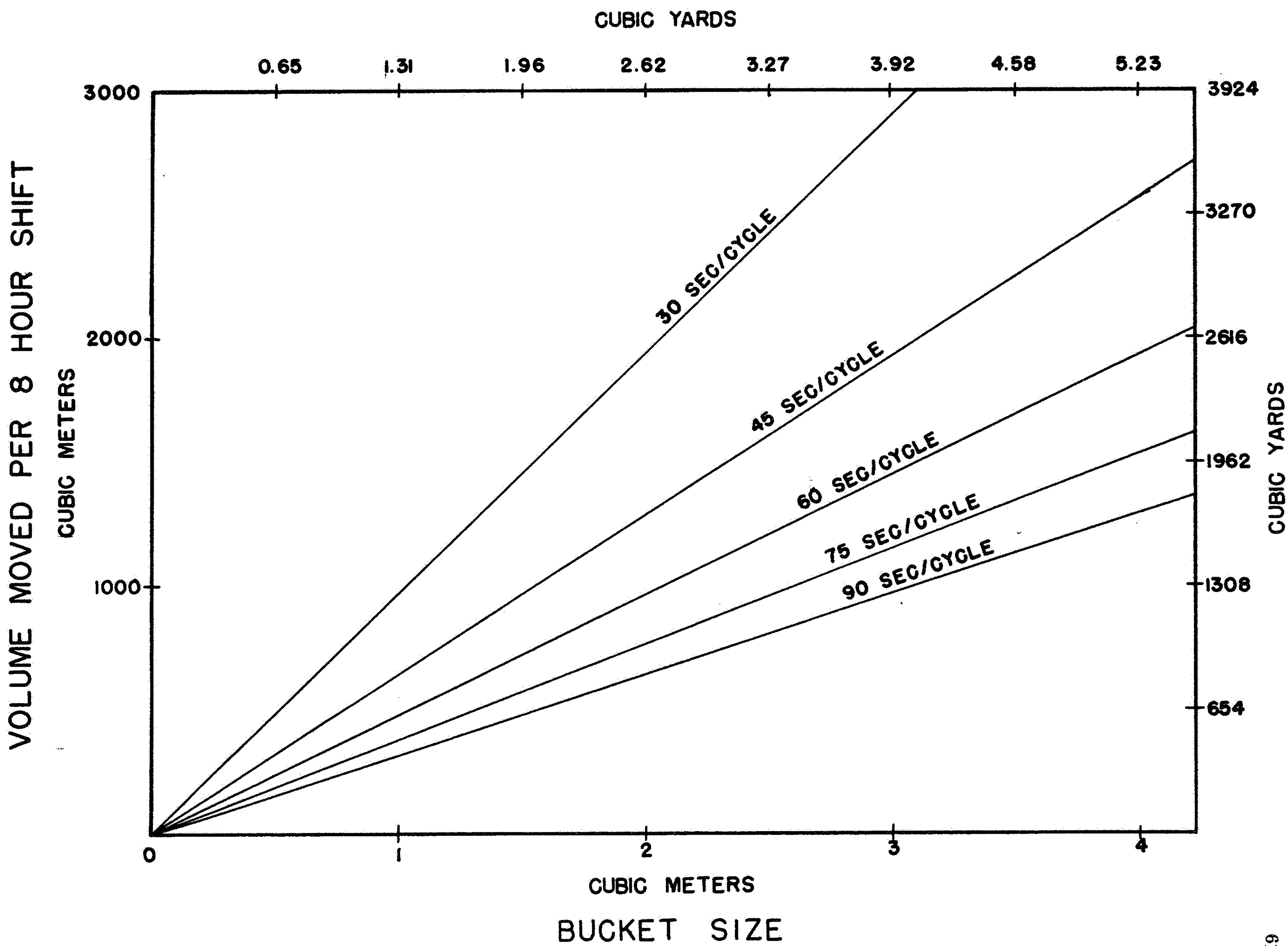


Fig. 10. Volume of material moved per 8-hour shift by a shovel or dragline, as a function of bucket size for 5 cycle lengths.

atmospheric damping, may result in a wider amplitude of swing of the bucket over the dump point and tend to prolong the cycle. Thus, draglines may have longer cycle times than shovels of equivalent capacity but 60 seconds per cycle for both types of machine, has been used.

The weight reduction formula used for shovels and draglines is: 0.20 (steel or equivalent density material) + 0.65 (Ti-6Al-4V, alloy substitute for steel) + 0.15 (aluminum) = 0.62. This factor gives the maximum weight reduction calculated for any kind of equipment. Aluminum may be used in the end sections of booms and in cabs.

Table 5 shows that all draglines and shovels are considerably too greatly oversized for deposit No. 2 and grossly oversized for No. 1. This is in spite of the fact that the smallest models advertised by three prominent manufacturers(1) were selected.

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(1) Bucyrus-Erie Company, Manitowac Engineering Company and Marion Power Shovel Company.

Although this type of equipment should be usable on all lunar water deposits, for the size deposit chosen it has tremendous economic disadvantages. Some of these are:

(1) It could not be determined, in most cases, if the weights quoted included counterweights. The magnitudes of counterweights were given in some cases and if these were included, considerable saving could be made by not shipping these and using indigeneous lunar material.

(2) All units were so grossly oversized that all required only the minimum crew. The necessity to transport a heavy stand-by machine to the moon, for each heavy productive one, however, more than offset this advantage. A previous report 49/ in which no provision for a stand-by was made, indicated the smallest shovels

and draglines are competitive with other mining systems. The per-shift costs shown in Table 5, when compared to those of Tables 1-4, do not indicate this.

Shovels and draglines on earth are rugged, long-lived machines. Providing a stand-by machine could be off-set by spreading the transport charge over a 2-year or 360-shift period, if this can be justified. The results of such a procedure are shown in Table 6.

Table 6

Shovel and Dragline costs/shift with transport costs charged to 360 shifts

Machine No.		System Weight		Costs Per Shift, $	
		kg	Lbs	Transport	Total
1	(Shovel)	62,230	137,190	1.91×10^6	4.31×10^6
2		93,350	205,800	2.86×10^6	5.26×10^6
3		114,490	252,400	3.51×10^6	5.91×10^6
4		99,280	218,860	3.04×10^6	5.44×10^6
5		113,900	251,100	3.49×10^6	5.89×10^6
1	(Dragline)	67,480	148,760	2.07×10^6	4.47×10^6
2		83,620	184,340	2.56×10^6	4.96×10^6
3		96,410	212,550	2.95×10^6	5.35×10^6
4		101,240	223,200	3.10×10^6	5.50×10^6
5		104,620	230,640	3.20×10^6	5.60×10^6
6		108,560	239,320	3.32×10^6	5.72×10^6
7		117,850	259,800	3.61×10^6	6.01×10^6
8		51,930	114,480	1.59×10^6	3.99×10^6
9		116,830	257,560	3.58×10^6	5.98×10^6

Table 6 indicates much improvement over the costs of Table 5 but only shovel No. 1 and dragline No. 8 appear competitive with bulldozers (and only at bulldozer speed of 1.93 km/hr), front-end loaders and cable excavation systems. They are not competitive with scraper-loader systems but they can be used under all rock hardness and toughness conditions whereas this is not true of the other systems.

G. Comparative first-order costs of six systems: If the assumptions upon which the costs shown in Tables 1-5 were calculated have reasonable validity and the selection of equipment models is representative, the costs should be within one order of magnitude of those that may eventually be found. Even if the spread proves greater, the costs shown should be fairly accurate, relative to each other.

The real competition to any lunar mining venture is the cost of the same product from the earth, delivered to the moon. If this cost proves to be $5,000 per pound of water, all lunar water mining ventures must be measured against this figure. Water from the earth probably will not be delivered to the moon, as such, but in the form of LOX and LH_2. In either case, if the proportion of these two is near 8:1, by weight, the delivered mass will be the same.

Shipment from the earth to the moon will not be quite the total cost of earth water on the moon but the $5,000 per pound transport cost will be so much larger than others that, in a first order study, all other costs can be neglected.

In the calculations for Tables 1-5, net water to be produced is 122,728 kg (270,000 lbs) per year. Per shift this is 1/180 of 270,000 lbs or 1500 lbs (680.4 kg). At $5,000 per pound it will cost $\$7.50\text{x}10^6$/shift to place this quantity of water on the moon.

This does not mean that all mining costs in Tables 1-5, that fall below $\$7.50\text{x}10^6$, will be acceptable. Costs of water processing, lunar surface transport, energy and many other costs, must be allowed for. In the absence of estimates of these costs, it was decided arbitrarily to define any mining cost at or below 53.3% of $\$7.50 \times 10^6$, or $\$4.00 \times 10^6$, as an acceptable one and any cost between this figure and 71.3% of $\$7.50 \times 10^6$ or $\$5.35 \times 10^6$, as a borderline cost. Of course, all costs above $\$5.35 \times 10^6$ per shift will be defined as excessive.

The nearest thing to a study of processing costs available is that of Rosenberg, et al 50 /, who estimated the cost of extracting oxygen from rock by the Aerojet carbothermal process. It is practically certain that if water is present on the moon, processing oxygen from it will be cheaper than processing rock by the carbothermal process. Carbothermal process costs, then, probably can serve as a maximum limit for water processing costs.

Cost of plant delivery, plus cost of labor, is $\$387 \times 10^6$ for 144,000 lbs of oxygen per year and $\$463 \times 10^6$ for 288,000 lbs 50 /. The larger plant will produce oxygen at the lower cost per unit mass. Interpolating between the two figures, gives a cost of $\$438 \times 10^6$ for 240,000 lbs oxygen per year. This figure gives a cost of $1826 per pound. If the cost of water recovery, on a weight basis, is the same, the equivalent cost for water is $1826 \times 9/8 = \$2054$/lb. This figure is 41.8% of $5,000 and leaves 58.2% for mining and other costs. This percentage is close to the 53.3% ($\$4.00 \times 10^6$/shift) of water transportation costs that was assumed. If studies later show the cost of water recovery to be substantially below that of equivalent oxygen by the carbothermal process, the mining cost limits set here should hold up; if not, they may have to be revised downward from $\$4.00 \times 10^6$/shift.

Table 7 shows the costs for all the equipment alternatives of the six systems classified as "acceptable," "borderline" and "excessive," on the basis outlined above.

Only three of six equipment alternatives in the cable excavator system had sufficient capacity to mine deposit No. 1, at the slow dragging speed, and only one

Table 7

Table showing mining systems and equipment proposed for deposits No. 1 and No. 2 with costs classified as excessive (Ex), borderline (Bl) and acceptable (Ac). Costs ratios are shown, relative to the lowest cost = 1 (only for cost of $\$5.35\text{x}10^6$/shift, or less).

Mining System	Equip. No.	Deposit No. 1 Cost, \$	Ex	Bl	Ac	Deposit No. 2 Cost	Ex	Bl	Ac	Cost Ratios Dep. No. 1	Cost Ratios Dep. No. 2
					At 103 trips 18 hours						
Cable excavator	1	(1)	-	-	-	(1)	-	-	-		
	2	$4.70\text{x}10^6$		✓		(1)	-	-	-	1.53	
	3	(1)	-	-	-	(1)	-	-	-		
	4	(1)	-	-	-	(1)	-	-	-		
	5	$4.71\text{x}10^6$		✓		(1)	-	-	-	1.53	
	6	$4.88\text{x}10^6$		✓		$5.78\text{x}10^6$				1.59	
Bulldozers	1	$5.67\text{x}10^6$	✓			$11.51\text{x}10^6$	✓				
	2	$5.81\text{x}10^6$	✓			$11.83\text{x}10^6$	✓				
	3	$4.97\text{x}10^6$		✓		$9.38\text{x}10^6$	✓			1.62	
	4	$4.12\text{x}10^6$		✓		$7.75\text{x}10^6$	✓			1.34	
	5	$4.14\text{x}10^6$		✓		$7.81\text{x}10^6$	✓			1.35	
	6	$4.14\text{x}10^6$		✓		$6.38\text{x}10^6$	✓			1.35	
	7	$4.49\text{x}10^6$		✓		$5.75\text{x}10^6$	✓			1.46	
Front-end loaders	1(2)	$3.51\text{x}10^6$			✓	$5.52\text{x}10^6$	✓			1.14	
	2(2)	$3.78\text{x}10^6$			✓	$5.77\text{x}10^6$	✓			1.23	
	3	$4.84\text{x}10^6$		✓		$8.09\text{x}10^6$	✓			1.58	
	4(2)	$3.84\text{x}10^6$			✓	$4.85\text{x}10^6$		✓		1.25	1.47
	5	$3.95\text{x}10^6$			✓	$7.35\text{x}10^6$	✓			1.29	
	6	$4.09\text{x}10^6$		✓		$6.28\text{x}10^6$	✓			1.33	
	7(2)	$3.07\text{x}10^6$			✓	$5.34\text{x}10^6$		✓		1.00	1.62
	8(2)	$3.08\text{x}10^6$			✓	$4.22\text{x}10^6$		✓		1.00	1.28
	9	$4.25\text{x}10^6$		✓		$6.55\text{x}10^6$	✓			1.38	
	10(2)	$3.18\text{x}10^6$			✓	$5.57\text{x}10^6$	✓			1.04	
	11	$4.45\text{x}10^6$		✓		$5.66\text{x}10^6$	✓			1.45	
	12	$4.49\text{x}10^6$		✓		$5.72\text{x}10^6$	✓			1.46	
Scraper-loaders	1	$3.29\text{x}10^6$			✓	$3.29\text{x}10^6$			✓	1.07	1.00
	2	$3.32\text{x}10^6$			✓	$3.32\text{x}10^6$			✓	1.08	1.01
	3	$3.97\text{x}10^6$			✓	$3.97\text{x}10^6$			✓	1.29	1.21
					At 446 trips 18 hours						
Cable excavator	1	$4.56\text{x}10^6$		✓		$5.47\text{x}10^6$	✓			1.76	
	2	$4.70\text{x}10^6$		✓		$5.60\text{x}10^6$	✓			1.81	
	3	$4.55\text{x}10^6$		✓		(1)	-	-	-	1.76	
	4	$4.66\text{x}10^6$		✓		$5.56\text{x}10^6$	✓			1.80	
	5	$4.71\text{x}10^6$		✓		$5.62\text{x}10^6$	✓			1.82	
	6	$4.88\text{x}10^6$		✓		$5.78\text{x}10^6$	✓			1.88	
Bulldozers	1	$2.75\text{x}10^6$			✓	$4.69\text{x}10^6$		✓		1.06	
	2	$2.80\text{x}10^6$			✓	$4.81\text{x}10^6$		✓		1.08	
	3	$2.80\text{x}10^6$			✓	$3.93\text{x}10^6$			✓	1.08	1.52
	4	$3.01\text{x}10^6$			✓	$3.81\text{x}10^6$			✓	1.16	1.47
	5	$3.03\text{x}10^6$			✓	$3.83\text{x}10^6$			✓	1.17	1.48
	6	$3.03\text{x}10^6$			✓	$3.83\text{x}10^6$			✓	1.17	1.48
	7	$3.26\text{x}10^6$			✓	$3.26\text{x}10^6$			✓	1.26	1.48
Front-end loading	1	$2.59\text{x}10^6$			✓	$2.59\text{x}10^6$			✓	1.00	1.00
	2	$2.76\text{x}10^6$			✓	$2.76\text{x}10^6$			✓	1.07	1.07
	3	$2.79\text{x}10^6$			✓	$2.98\text{x}10^6$			✓	1.08	1.15
	4	$2.79\text{x}10^6$			✓	$2.79\text{x}10^6$			✓	1.08	1.08
	5	$2.86\text{x}10^6$			✓	$3.01\text{x}10^6$			✓	1.10	1.16
	6	$2.95\text{x}10^6$			✓	$2.95\text{x}10^6$			✓	1.14	1.14
	7	$3.01\text{x}10^6$			✓	$3.01\text{x}10^6$			✓	1.16	1.16
	8	$3.02\text{x}10^6$			✓	$3.02\text{x}10^6$			✓	1.17	1.17
	9	$3.04\text{x}10^6$			✓	$3.04\text{x}10^6$			✓	1.17	1.17
	10	$3.12\text{x}10^6$			✓	$3.12\text{x}10^6$			✓	1.20	1.20
	11	$3.16\text{x}10^6$			✓	$3.16\text{x}10^6$			✓	1.22	1.22
	12	$3.19\text{x}10^6$			✓	$3.19\text{x}10^6$			✓	1.23	1.23
Scraper-loaders		Same cost as at lower speed									

Shovels	1	$6.21\text{x}10^6$	✓			$6.21\text{x}10^6$	✓				
	2	$8.12\text{x}10^6$	✓			$8.12\text{x}10^6$	✓				
	3	$9.41\text{x}10^6$	✓			$9.41\text{x}10^6$	✓				
	4	$8.48\text{x}10^6$	✓			$8.48\text{x}10^6$	✓				
	5	$10.00\text{x}10^6$	✓			$9.38\text{x}10^6$	✓				
Draglines	1	$6.53\text{x}10^6$	✓			$6.53\text{x}10^6$	✓				
	2	$7.52\text{x}10^6$	✓			$7.52\text{x}10^6$	✓				
	3	$8.30\text{x}10^6$	✓			$8.30\text{x}10^6$	✓				
	4	$8.60\text{x}10^6$	✓			$8.60\text{x}10^6$	✓				
	5	$8.81\text{x}10^6$	✓			$8.81\text{x}10^6$	✓				
	6	$9.05\text{x}10^6$	✓			$9.05\text{x}10^6$	✓				
	7	$9.62\text{x}10^6$	✓			$9.62\text{x}10^6$	✓				
	8	$5.59\text{x}10^6$	✓			$5.59\text{x}10^6$	✓				
	9	$9.55\text{x}10^6$	✓			$9.55\text{x}10^6$	✓				

Footnotes:

(1) Equipment capacity too low to mine the deposit

(2) Wheeled models

could mine deposit No. 2. The costs are all borderline for deposit No. 1 and excessive for No. 2. Only the two smaller bulldozers show excessive costs for the thinner overburden but all costs are excessive for all bulldozers mining deposit No. 2 at slow pushing speed. Front-end loaders, often with the same tractors, have lower costs. Seven models (6 are wheeled) have acceptable costs for deposit No. 1, and the other five are borderline. For deposit No. 2, 3 models are borderline (all wheeled) and nine have excessive costs at slow speed. All three scraper-loaders have acceptable costs for mining both deposits.

At the faster speed assumed, all systems show more favorable costs, as expected. All units of the cable excavator system have sufficient capacity to mine deposit No. 1, and all except one, to mine deposit No. 2. Costs are still not favorable, all being borderline for No. 1 and excessive for No. 2.

As in the case of shovels and draglines, if the heavy head and tail towers can be charged to more than 180 shifts, which appears reasonable, costs of the cable excavator system can be lowered considerably.

All bulldozers have acceptable costs for deposit No. 1 and all except the two lightest, for deposit No. 2, at the higher speed. The two exceptions are borderline. All front-end loaders have acceptable costs for both deposits at a mining rate of 446 trips/shift.

All models of shovels and draglines have excessive costs for both deposits. They are all simply too oversized for even the larger deposit. The smallest dragline comes close to borderline cost rating for deposit No. 2 but among shovels, only No. 1 even comes under the cost of water transport from the earth. Table 6 shows the beneficial effects of charging the transport costs of durable, heavy equipment to 360 shifts rather than to 180 shifts. This change lowers mining costs for two shovels and 3 draglines to borderline or acceptable ranges, with dragline No. 8

costs being the only one that is favorable. Shovel No. 1 is only slightly behind dragline No. 8 in economy.

The last two columns of Table 7 show the costs of all equipment alternatives and systems not having excessive costs ($\$5.35 \times 10^6$), relative to the lowest cost equipment and system at the particular mining speed or rate and for deposits No. 1 and No. 2. Front-end loader No. 7, with a cost of $\$3.07 \times 10^6$ per shift, is the lowest for deposit No. 1 and 103 trips per shift. Scraper-loader No. 1 is the lowest for deposit No. 2 for the same number of trips, at $\$3.29 \times 10^6$ per shift. At 446 trips per shift (high speed for all scraping methods), front-end loader No. 1, at $\$2.59 \times 10^6$ per shift, is the lowest cost for either deposit.

For mining deposit No. 1 at the low rate, 2 of 12 front-end loaders (both wheeled models) and 2 of 3 scraper-loaders, are within 10% of the lowest cost; 5 front-end loaders, all wheeled, and 2 scraper-loaders are within 25%; and 4 of 7 bulldozers, all front-end loaders and all scraper-loaders, are within 50% of the lowest cost equipment. All other equipment and systems with costs under $\$5.35 \times 10^6$ per shift are more than 50% higher than the minimum cost.

For deposit No. 2, one scraper-loader comes within 10%, the third scraper-loader within 25% and wheeled front-end loaders, No. 4 and 8, within 50% of the lowest cost equipment, at 103 trips per shift.

At faster speeds, giving 446 trips per shift, bulldozers No. 1 - 3 and front-end loaders No. 2 - 5 have costs within 10% of that of front-end loader No. 1; bulldozers 4 - 6 and front-end loaders 6 - 12 are within 25%, and only bulldozer No. 7, at 26%, exceeds 25% of the lowest cost equipment for mining deposit No. 1. The spread is much wider for mining deposit No. 2 at higher rates. Bulldozer No. 3 costs exceed the minimum one by more than 50% and bulldozers Nos. 4 - 7 are just less than 50% above this cost. No front-end loader exceeds the minimum by as much as 25% and Nos. 2 and 4 are less than 10% above it.

On the basis of the figures in Table 7, all shovels and draglines are so oversized that the only way they can be competitive with the systems using small equipment is to (1) risk not sending an entire spare unit or (2) charge transport costs to more than 180 shifts.

Cable excavator systems, to be useful, must have sufficient size buckets or scrapers to mine the deposit. Regardless of bucket capacity or speed of scraping, however, most of the system weight is in the towers and the only way to make the system competitive is to (1) design much lighter towers or (2) charge the transport costs of the towers to more than 180 shifts.

The reason for high or low costs among systems is controlled largely by their weight-to-capacity ratios. Table 8 shows these ratios for all the machines of sufficient capacity to mine deposit No. 2 at low speed.

The ratio variation is quite wide. This is especially true of bulldozers where ratios vary from 232 to 933. Front-end loaders have ratios as low as 94 and as high as 430.

Fig. 11 shows the ratios of Table 8 plotted against the corresponding costs from Table 1-5. Points representing different equipment models in the same system plot as straight lines with little scatter. Scatter is caused by inherent differences due to machine design and to variation in personnel costs.

Machines of a given system, exactly tailored for a given deposit, should plot similarly to the bulldozer and front-end loader lines of Fig. 11. The more units that are needed (undersized, individual units), the lower the slope of the line. Oversized units give a vertical line (shovels) or a line with a negative slope (scraper-loaders and draglines). Draglines fall near two separate but parallel lines rather than near a single one, probably due to the fortuitous choice of models. Fig. 11 also shows, graphically, that scraper-loaders, wheeled front-end loaders and the

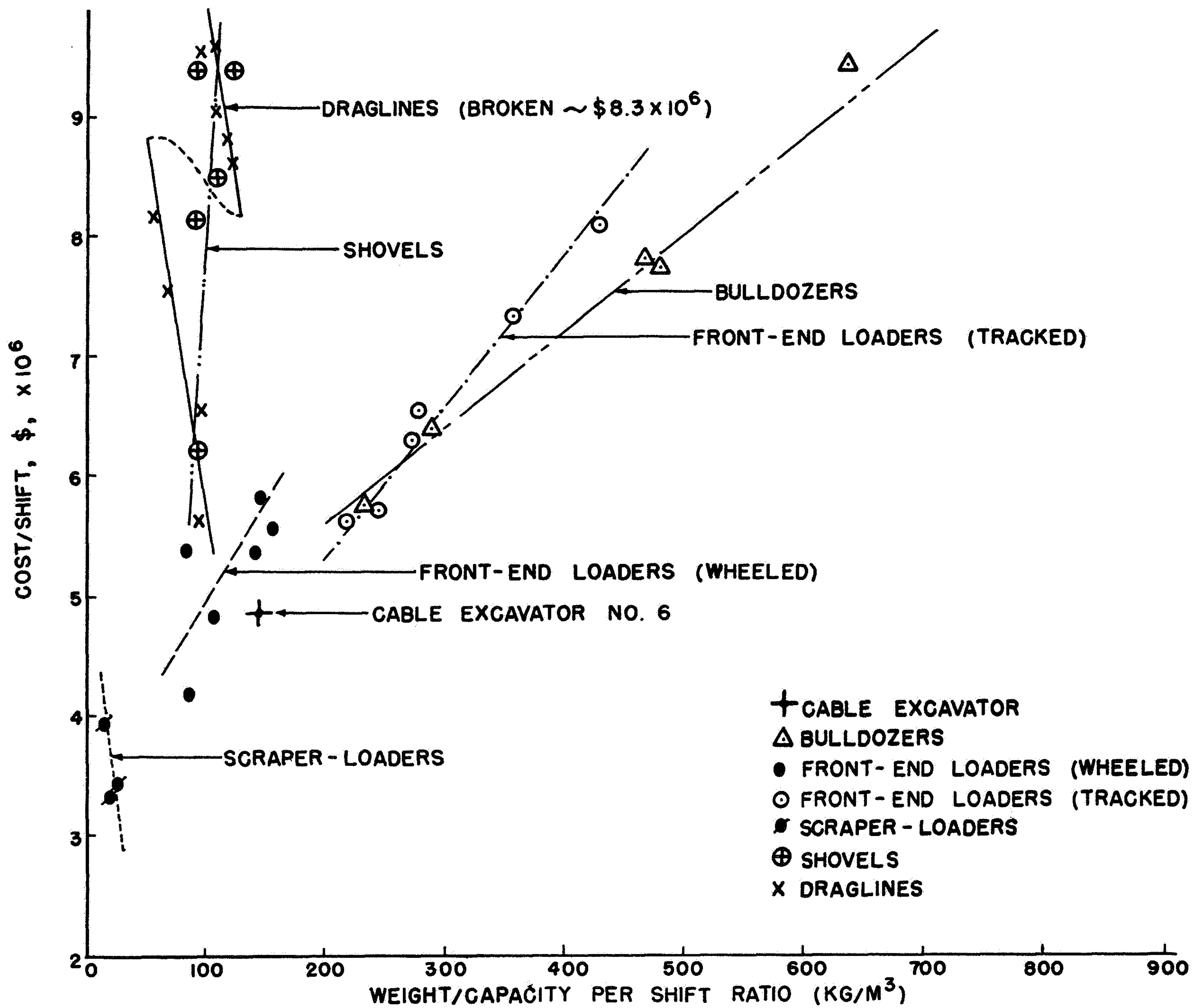


Fig. 11. Plot of cost per shift of various mining equipment systems against the ratio of the system weight to its volumetric capacity per shift.

Table 8

Table showing ratio of system weight to mining capacity, per shift, for deposit No. 2, at the lower speed.

Mining System	Equip. No.	Ratio, System wt. (kg) to Capacity (M^3)	Mining System	Equip. No.	Ratio, System wt. (kg) to Capacity (M^3)
1. Cable				10	162
excavator				11	220
	6	141		12	238
2. Bulldozers	1	800	4. Scraper-	1	26
	2	933	loaders	2	25
	3	641		3	23
	4	481	5. Shovels	1	90
	5	473		2	90
	6	288		3	89
	7	232		4	108
3. Front-end	1	82		5	124
loaders	2	145	6. Draglines	1	98
	3	430		2	70
	4	107		3	57
	5	363		4	123
	6	272		5	120
	7	139		6	113
	8	94		7	112
	9	277		8	94
				9	91

one cable excavator model, are inherently low cost systems, if they are properly fitted to the production rate of the deposit and can be used. Bulldozers, and tracked front-end loaders, can be competitive only with high capacity models. Surprisingly, shovels and draglines can be competitive also but they must not be grossly oversized for the production rate. If weight per unit productivity (not per unit capacity, which may be too high) is low, any mining system probably can be designed to yield costs under $5.35x10^6/shift, with the assumptions made in this report.

H. General Conclusions: The smaller machines, such as bulldozers and front-end loaders, are more economical than oversized machines, like draglines and shovels, especially for mining shallow deposits. If machines are oversized, they are uneconomic to transport from the earth; if undersized, manpower costs go up sharply because of the number of units needed.

2. Scraper-loaders, if usable, appear to have the best promise of economic operation. Wheeled front-end loaders are second. If either of these systems will work, short distance haulage problems may be solved also because both haul as well as scrape or dig and load.

3. If lunar rock and hydrous ore characteristics are such as to exclude digging and scraping directly, properly sized shovels and draglines should be competitive. After fragmentation of rock, it may be scrapable with cable excavator, bulldozer or dragline or diggable by front-end loader or power shovel. The latter should be most certain to handle the material if there is much vacuum adhesion.

4. Cable excavator systems should be most conveniently mechanized, as the term is used in mining. Operation mechanically from a single shelter is entirely possible. All other systems will almost certainly need an operator on the machine, or machines, in or near the pit.

5. High capacity units must be used to secure low weight/capacity ratios and minimum sized operating crews.

6. To secure sufficient capacity, small scrapers, bulldozers and front-end loaders must possess considerable speed. While the low speed assumed in the calculations may be realizable, it is doubtful if the high speed assumed is possible to achieve in scraping operations but may be reached in transport to dumps, bins or storage piles. Crawler models are less likely to achieve the higher speeds than are wheeled ones.

3. <u>Other Problems and Costs, if</u> - A. Fragmentation of overburden and/or ore, is required: The costs of drilling, blasting or other method of fragmentation, must be added to those of scraping or digging, part or all of the material in the pit,

if the ore proves too hard or too tough to scrape or dig directly. In the case of material that is difficult, but possible, to scrape or dig, the total cost of the one difficult operation should still be less than the sum of the costs of two tandem operations although the scraping or digging will be faster, easier, and less destructive of blades or teeth after fragmentation than without it. If scraping or digging is not too difficult, therefore, prior fragmentation probably will be omitted. In the case of solid rock of anything near earth-like properties, fragmentation will be required and will add to mining costs.

If ore containing 2% water must be mined at the rate of 37,879 kg (83,333 lb) per shift, the water produced will weigh 758 kg (1667 lb). Interpolating in Table 10, 0.0039 kg by explosives will be required for each kg of water or 2.96 kg of explosives per shift for "hard" ore. The cost of transportation from earth will be $32,500, a really negligable addition to excavation costs.

While the cost of transportation of explosives, drills, and drilling supplies from the earth will be considerable, it is probable that the greater part of the cost added by fragmentation will be in the cost of drilling. Even if the drill is made entirely automatic so that it can be operated by one man, the operations of drilling, loading holes and firing will require a minimum of a two man crew. The crew cost will then be $\$1.6\text{x}10^6$/shift. If this is added to the cost of front-end loader No. 7, Table 3, for deposit No. 1 at low speed, the total cost of mining becomes $\$4.67\text{x}10^6$/shift and places the system cost into the borderline class, out of the acceptable one.

It may be noted that this additional labor cost must be incurred by any underground mining system, other than with a tunneling machine although it may be minimized with forms of block caving.

B. <u>Mineral Deposits are too deep for Surface Mining</u>: If lunar water deposits are found only by drilling to a depth too great for surface mining, some method of underground mining will be required. As is the case on the earth, economic factors will determine the depth beyond which surface mining is impractical. After this has been determined, advances in surface mining technology may increase that depth in a short time.

Based on earth coal mining experience, about 150 feet (45.7m) is the present tentative limit although little mining at depths greater than 100 ft (30.5m) is actually being done. It may be argued that, because of differences in gravitational attraction, surface mining on the moon can be done at 6 times this depth, or about 900 ft (274.4m) with present large earth machines. As has been pointed out, however, lifting against gravity may not be the most difficult part of the operation and the practical lunar limit will be much less than 900 ft. Furthermore, present large earth machines are nowhere near economic for the anticipated lunar water mining demand (IV 2) and lower capacity machines must be used.

Hypothetical deposit No. 6, with 100 m (328 ft) overburden, probably will be too deep to mine with the small shovels and draglines of Table 5, although all these were oversized for the 30 m of overburden over deposit No. 2. Without additional study and calculations, it may tentatively be concluded that more than 100 m of scrapable overburden will be uneconomic to remove for the water requirement postulated for 1982. If the overburden is solid rock, requiring fragmentation prior to loading, this limit may be reduced to 30-50 meters (98.4 - 164.0 ft).

Ore mining by drilling, as suggested in a previous contract report 1/, will also be uneconomic at 1982 oxygen demands. This means that indirect systems, similar to the Frasch process for sulfur 5,38,39/, or some form of stoping, after reaching the deposit via vertical shaft or slope, will be required. No extensive study of such methods was made on this contract but a brief discussion was included in a previous report 1/.

4. Probable Adaptability of Mining Systems to the Seven Deposit Models: It was assumed in Section IV 2 that deposits No. 1 and No. 2 can be mined by any or all the six systems discussed. In addition, the tractor drawn scraper and wheel excavator could mine deposits mineable by scraping methods and requiring no prior fragmentation.

It is possible, of course, that overburden deeper than about one meter 14/ and ore in deposits No. 1 and 2, may prove too tough for scraping or digging without fragmentation. In general, if this is true, both can still be mined by all methods, after fragmentation. If the overburden contains many large boulders, or meteoritic iron masses that do not disintegrate under light blows, scraping methods and digging methods, especially the latter, will not be suitable. Mining costs will go up sharply if many such masses must be handled. In the area on which Surveyor I landed, the surface boulders do not appear to be too large for a one cubic yard or larger front-end loader or shovel bucket to handle and a bulldozer should be able to "wrestle" most of them out of a pit 14/. Furthermore, some of the rock masses appear to have broken on landing on the lunar surface 14/ after ejection from a crater but unfortunately it is not possible to estimate how hard they impacted. If boulders too large to handle are not numerous, they should have little effect on the mining operation except to slow down the rate of excavation. If fragmentation, prior to scraping and loading, is practiced, all objectionable masses should be sufficiently shattered for handling.

Summarizing, deposits No. 1 and No. 2 may be mined by any of the systems discussed, the choice of system depending ultimately upon the physical hardness, toughness and size consist (if not solid rock) of the materials to be handled.

Deposit No. 3 overburden possibly may be mined by all systems. Crater rims may have some very large blocks embedded in them but these should be rather small in the 100 m diameter crater of deposit No. 3. In much larger craters, blocks may be so large as to eliminate scraping and digging methods. Even though the ore zone is a breccia in deposit No. 3, it will require fragmentation before loading if it is cemented or if the blocks are large. Blockly material, even if not firmly cemented, will be difficult to scrape. Small but hard blocks will be difficult to dig. After fine fragmentation of the ore, which will require a considerable explosive charge, it can be picked up by any of the machines. If fragmentation is coarse, a power shovel, cable excavator or a dragline system probably will be best for excavation.

What has been said concerning mining of the ore in deposit No. 3 can be repeated for the ore in deposit No. 4. The ore in No. 4 will require as much or more fragmentation than that from No. 3. The thin overburden may be scrapable but the hanging wall rock can range from solid basalt to a triable tuff, depending on its composition and origin or it may have a fragile, vesicular or "cotton candy" structure, especially near the top, if it is a lava extruded into vacuum. The ore will, of course, have essentially the same composition as the mare filling (except for the minerals containing the water) but a shear or facture zone will possess one or more planes of comparatively easy separation. An expanded, pumice-like structure could make both ore and rock scrapable and possibly digable but if that is its condition, its density will be much less than that assumed in the model. Deposit No. 4, as specified, probably will be most readily mined by the three methods suggested for mining the ore of deposit No. 3.

The ore of deposit No. 5 should be scrapable unless it is cemented by mineralization or contains too many large blocks. The same is true of the 75 m of unmineralized rubble over the ore. The hangingwall rock, however, probably must be fragmented. If fragmented finely enough, it can be scraped and dug because it will be blasted into the rill and not fragmented in place. Benching the material in the hangingwall of deposits 4 and 5 will be easier than for the breccia in deposit No. 3, especially when working toward the center of the latter deposit. Scraping and digging systems probably can be used to mine deposit No. 5.

Deposit No. 6 presents sharply contrasting properties for overburden and ore. The overburden can be scraped or dug. The ore will require fragmentation and if coarsely fragmented, may be difficult to scrape or dig for loading. The ore is almost certain to be solid because the weight of the material under which the laccolith was intruded prevented the distention that would occur with extrusion to the surface.

The deposit might be mined by two crews: one scraping overburden ahead of ore extraction, using scraper-loaders, front-end loaders, bulldozers, shovels or draglines and the other mining ore by fragmentation and removal by shovel, dragline or tower excavator. With a serpentine ore body, uncovering ore well in advance of mining will not result in any dehydration of ore. The very thick overburden, 100 m, may require too many machines to be handled by bulldozers or front-end loaders. If the lower layers of overburden are baked hard from the heat of intrusion or cemented by contact metamorphism, some fragmentation may be required and all overburden may best be mined by power shovels.

There is one other possibility which was not mentioned in the description of deposit No. 6. The deposit could be a pseudo-laccolith which was extruded on the surface of the mare and later covered by a falls or flows of tuff material. In this case, at least the upper part would be distended and fragile and might be scraped. There would also be no contact metamorphic effects or consolidation of overburden.

As the model stands, power shovels or draglines will be the most probable mining system.

Model 7 represents a type of ore body ordinarily not mined by surface methods because of its narrowness. It is only 1.55 m (5.0 ft) wide and a small piece of equipment could hardly work inside the fissure containing the ore body. To mine it to any depth by the systems considered for deposit Nos. 1 and 2 would require removal of very much wall material of little value. It could be mined, however, by a ditching machine or a hoe. If the fissure material is soft or crusty, the machine would have to stand altogether on the side of the fissure. There is a definite depth limit to which any equipment model could mine. Mining below that limit would necessarily be by underground methods or by a surface system removing much, if not all, the material between fissures. If the deposit were of great linear extent or if a large area is mineralized like deposit No. 7, it might be practical to mine it to shallow depths only.

V

POWER FOR LUNAR SURFACE MINING

1. Possible Power Forms and Power Demand: Just as in the case of earth surface mining, the basic choice of a power system for lunar mining appears to be between individual units for each machine or operation and a central system with distribution through trailing cables. In either case, power will undoubtedly be electric 49, 51/. Heat engines, either internal or external, are unsuitable for lunar conditions 51, 52/, and their fuels must be transported from earth unless they, too, can be found or manufactured on the moon.

To date, most of the published material on space power systems has dealt primarily with power for space craft 51, 53, 54, 55/, with some references to early lunar landings. At least one paper discusses nuclear thermionic lunar base power 56/. Mining and processing operations near a lunar base would be certain to add substantially to power requirements and may, indeed, become the principal power load.

Fig. 12 51/ shows, diagramically, the various possible power sources and conversion devices. Chemical and solar systems have been used successfully on spacecraft but large power requirements may depend upon nuclear or solar concentrator systems.

Fig. 13 51/, shows the approximate range of power levels, as related to lifetimes, for various space power systems. It will be noted that there is no lifetime limit on solar or nuclear systems but only the nuclear system has a high ($>\sim 10$ kw) power output. On the lunar surface, chemical-mechanical, fuel cells and batteries would not have the restricted time limits shown in Fig. 13 if (1) a continuous supply of fuel could be fed to a chemical-mechanical system, (2) fuel cells are made regenerative and (3) batteries are continuously or regularly recharged. Radioisotope

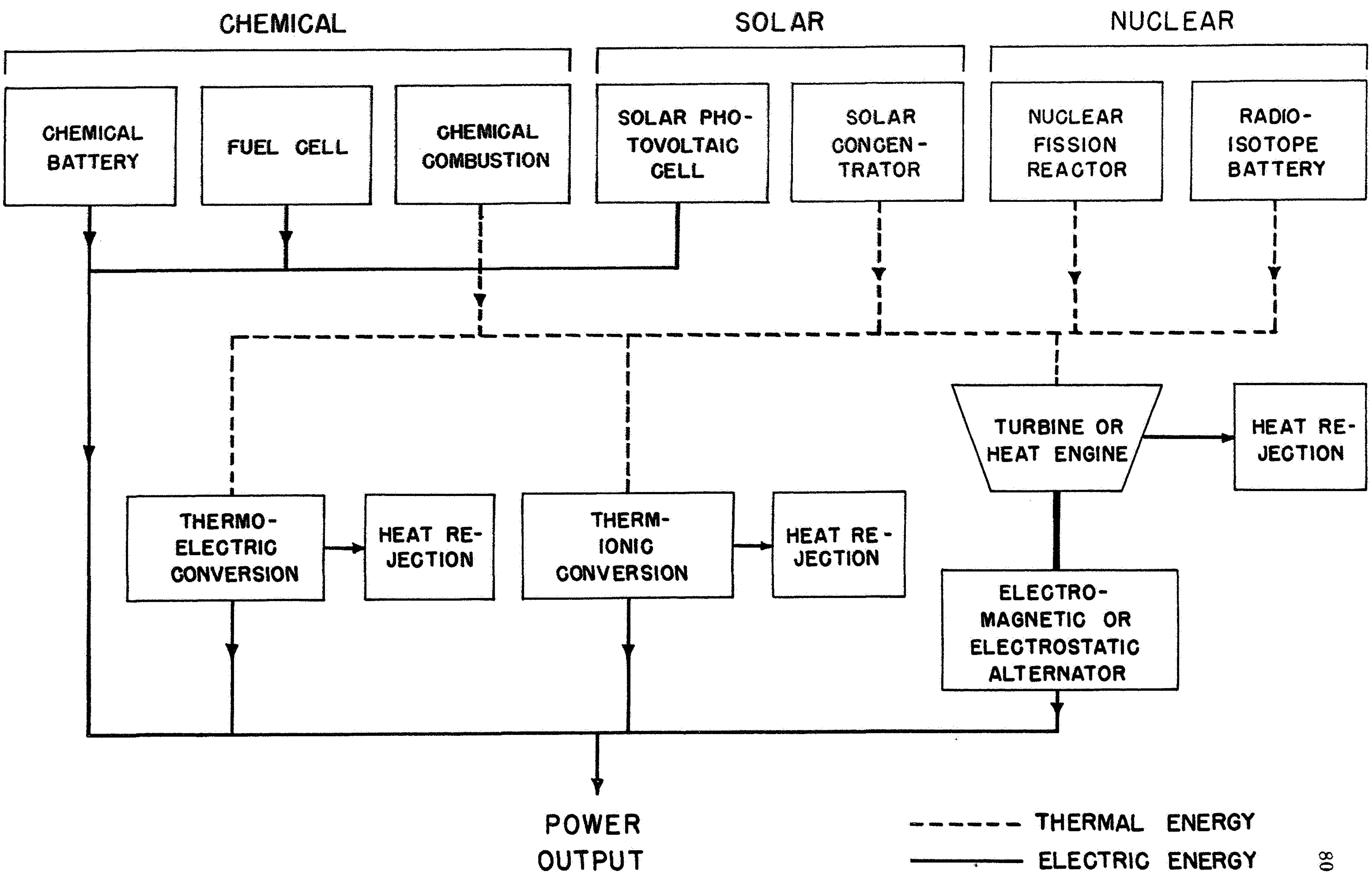


Fig. 12. Power sources and conversion devices. Dashed arrows - Thermal energy; solid arrows, electrical energy 51/.

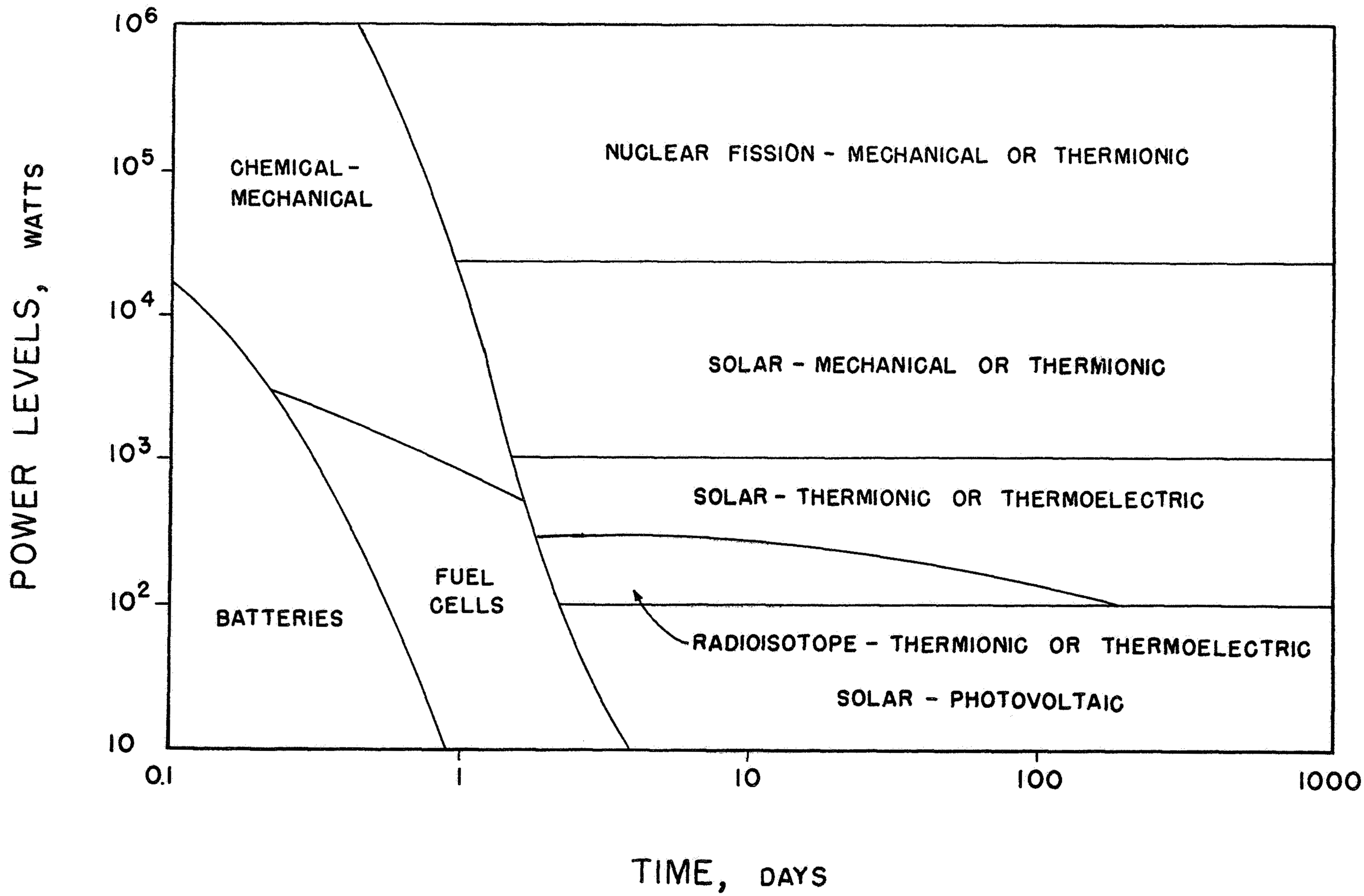


Fig. 13. Space power systems for various power levels and lifetimes 51 /.

systems would have approximately the lifetime limits shown, on the lunar surface. Bucher 51 / says: "It may be seen that for relatively low power levels and long lifetimes, the solar photovoltoic system is useful. For slightly higher power levels and moderate lifetimes, the nuclear radioisotope battery with thermoelectric or thermionic conversion is desirable. For longer lifetimes in approximately the same power range, solar concentrator systems with thermoelectric or thermionic conversion gain favor. At higher power levels, with moderate to long lifetimes, the solar mechanical or solar-thermionic systems become attractive. And finally, at the highest power levels the nuclear fission mechanical or the nuclear thermionic systems are most efficient."

The size of the power demand for mining is not easy to estimate. Looking at the motor horsepower ratings of the specific items of earth-based equipment considered in this study, to date, reveals a maximum rating of 186.5 kw (250 horsepower) on the largest power shovel. Modification of weight by incorporation of parts made from lightweight metals and alloys for lunar use should substantially reduce that requirement. In use on the moon, ballast or counterweights of local origin can be placed on equipment to add mass, if such addition is needed. Power required on the moon for lifting loads at earth speeds should be reduced to 1/6 of earth values. Locomotion requirements may not be reduced that much, however, if there is significant vacuum adhesion of lunar surface particles or if the bearing strength of the lunar surface in places, is low and allows considerable sinkage of treads or wheels.

Probably the governing factor on power requirements for lunar mining equipment will be that required for digging. If lunar rock or soil is as tough and adherent as is earth soil and rock, power requirement at constant speed may be equal to those on earth. In any case, requirements should lie between 1/6 and full earth values.

From studies of Surveyor I photos and other data 14 /, it appears doubtful if there is enough lunar vacuum adhesion to require more digging power than on earth. If we select a compromise figure of one-half earth power, 250 earth horsepower are equivalent to 125 horsepower on the moon or 93,250 watts or 93.3 kw. The power requirement for presently planned surface mining operations (including no power for processing) might conservatively be set at 100 kw (134 HP). From Fig. 13, it is seen that 10^5 watt loads virtually rule out systems other than chemical-mechanical or nuclear fission. With enough concentrating mirrors, a solar concentration-mechanical system might be brought to this level also but its impotence during the lunar night would seem to make it impractical.

2. Power Source: Levedahl reports on the development of a 100-kw(e) nuclear-thermionic power plant 56/. He says: "In both the intermediate and advanced stages of (Apollo) operation, only nuclear power plants appear capable of providing the very large quantities of energy necessary, and these plants must fit within the weight, size and center-of-gravity limitations of the advanced Saturn and Lunar Logistics Vehicle. Of the several types of nuclear power plants which have been studied, the thermionic nuclear reactor system has many advantages which uniquely qualify it for lunar power application." As quasi-permanent bases have been suggested for as early as 1975 18 /, 1982 power requirements should be well beyond a single power plant of the capacity Levedahl discusses.

The suggested lunar installation of the power plant is shown in Fig. 14 56 /, in a lunar depression or crater. A suitable crater, requiring little or no modification, should be located a short to moderate distance from almost any mining area. A depth of 10 feet or more below the surface will protect men on the surface who are not too close to the nuclear plant. A raised crater edge will provide perfect shielding, in the absence of any atmosphere to reflect nuclear radiation to personnel

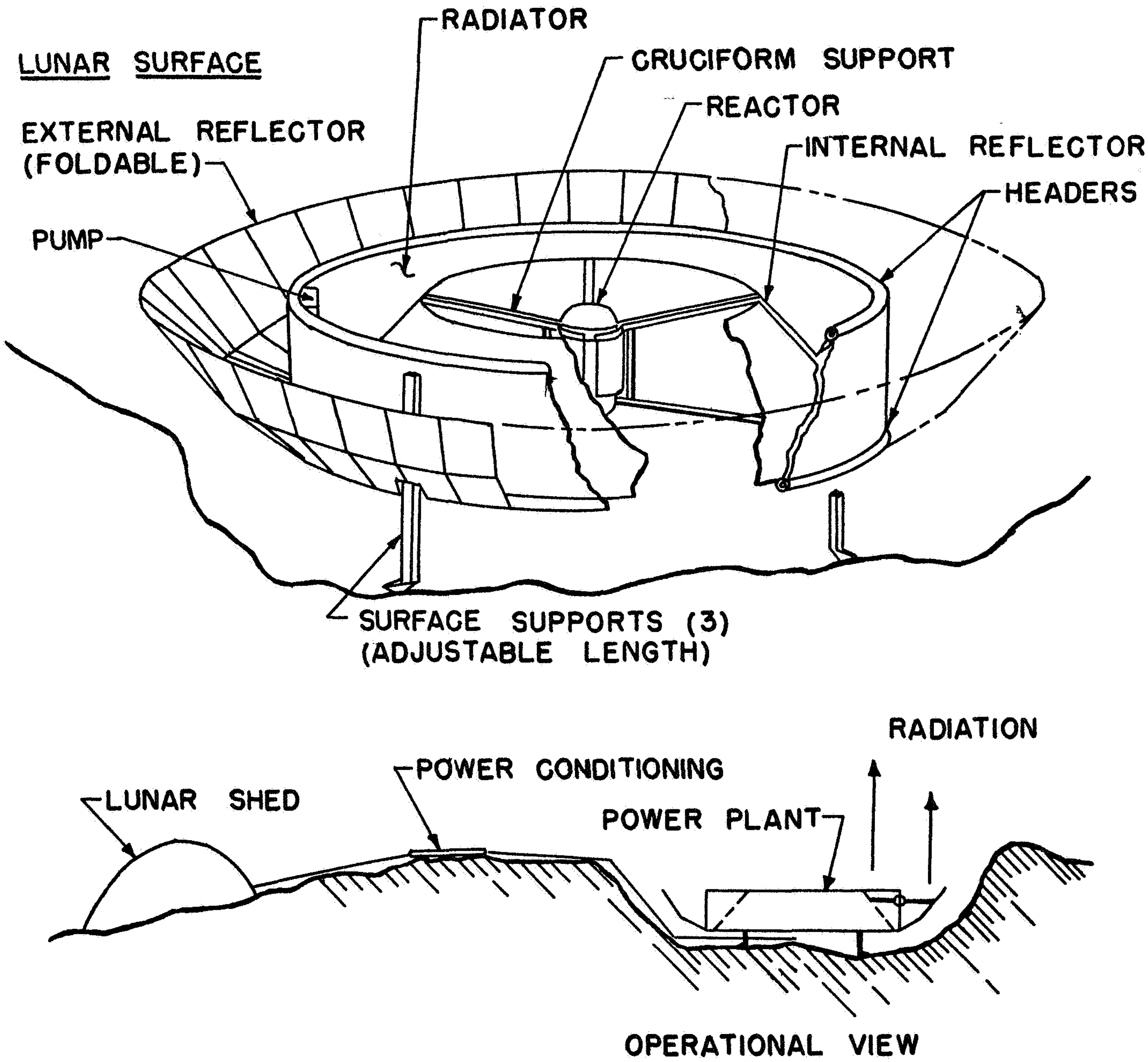


Fig. 14. Lunar power plant concept 56 /.

on the surface. In addition, if reflectors are placed on both the inside and outside surfaces of the radiator, heat rejection capacity is greatly increased, exceeding by 60% a single surface facing outer space 56 /.

The power conditioning system, as shown in Fig. 14 56 /, can be external to the power plant and can tailor frequency and voltage and/or convert to AC, as may be needed for mining operations. If the plant also powers a lunar base or a processing operation, requiring different power conditions, this unit, or units, can provide for both requirements. Because of the development of higher voltages, the flexibility provided and the possibility of incorporating d-c/a-c power converters into the power conditioning equipment, the thermionic nuclear power plant 56/ should prove the most satisfactory source for the type and quantity of power needed for the scale of mining operation envisioned.

It is beyond the scope of the current project to go any further into the power source problem but Levedahl suggests that such a system could be developed in about five years from go-ahead and at a cost, including fabrication of 2 or 3 operational systems, less than that of transportation of one lunar logistic vehicle to the surface of the moon 56 /.

3. AC or DC Power? Electrical power for earth-based surface mining operations is almost always alternating current. It may also prove desirable to use AC in lunar operations.

If individual power generators are used on mobile machines, however, it is almost certain that DC will be generated. "Currently our most developed schemes such as fuel cells generate direct current" 57 /. Bucher 51 / states that "One disadvantage of the fuel cell is the low voltage D-C output. For high voltages, cells may be placed in series, or d-c/a-c converters can be used." All chemical and solar systems, except chemical-combustion or solar concentrator systems, would have similar characteristics.

Although d-c power is probably used in all space vehicles to date 54 /, it is unlikely that type power will be satisfactory for mining operations on the scale being investigated. A representative of an electrical manufacturer writes: "In all probability, direct current motors will not be used because of the difficulty of providing an artificial environment for commutation and insulation effect. - - - one might envision a DC motor as having to be 'canned' in oil in order to operate in a deep space environment. The problems of sealing, connections and weight-to-power ratio would become obvious, using this technique." 52/. Another writes that: "Lifetime of motor brushes would pose a problem in the lunar vacuum" 58/. Work has been done on the development of brushless DC motors but "most of the present brushless DC motor designs for space vehicles are limited to output power levels of a few hundred watts" 54 /. Hope was expressed for improvement in design but application in space vehicles is still the use referred to. Some success has also been achieved in the development of long-life brushes for vacuum use 58 /.

A-C current apparently would be easier to handle and more flexible. "We have had some experience trying to make a small motor - - - using the approach of sealed, oil-cooled commutator-type d-c motor operating from a fuel cell. The approach may be satisfactory for very limited, short-time operations, but any major mining operation would require the use of an a-c motor" 57 /.

4. Central Distribution vs Individual Power Units: If lunar mining is done by drilling, requiring moving from one location to another 5 / or by the use of relatively small equipment units such as bulldozers or front-end loaders operating singly in small deposits or in multiples, on large ones 49 /, there might be some advantage in having individual power sources for each machine. If so, the fuel cell appears to be the most promising unit. It seems likely, however, that further development of this source is necessary before its use becomes possible. "Until

some larger power supply other than the Apollo-type fuel cell is available, motor ratings will be limited to 5 HP or less. A much more satisfactory system would use an inverter and an a-c motor, both of which would have to be hermetically sealed and cooled with a radiator and either circulating gas or oil. - - - motors of the larger sizes you mentioned would undoubtedly have to be a-c operated from a nuclear power plant - - -" 57/. Only the very smallest of bulldozers or front-end loaders investigated and the existance of very favorable lightness and looseness in lunar materials, would permit operation at 5 HP or less. One small rubber-tired front-end loader manufacturer advertises a 15.5 and a 24 HP (11.6 and 17.9 kw) model with a 1000 lb load capacity. With sand weighing 1760 kg/m^3 (110 lb/ft^3), this gives a volumetric capacity of only 0.26 m^3 (9.1 ft^3 or 1/3 yd^3). If lifting is the most difficult task in lunar mining operations, motors of 1.94 and 2.98 kw (2.6 and 4.0 HP) rating, respectively, should provide sufficient power. In this case, however, the machine would handle an entirely inadequate volume of ore and overburden per shift, for the size model deposit chosen for 1982 water demands 49/.

Although small tractors, propelled by individual power units, will be indespensible for the mobility required for lunar exploration and for interbase travel, it now appears that they will not be too useful as mining equipment unless considerable progress can be made in the development of higher capacity, efficient power sources.

This conclusion leaves fixed equipment powered by larger a-c motors or mobile equipment, also powered by a-c motors and with current supplied through the lunar surface or by trolly wires or tracks or by trailing cables. Supplying current through the lunar surface probably will be entirely too inefficient even though the lunar surface may be more conductive than the earth's. Tracks, monorails and trollys restrict mobility severely. Trailing, insulated conductors on the ground restrict radius of mobility, but provide excellent mobility within the designed radius.

In addition to problems connected with size and length of cable, there will also be problems of temperature and materials. At 104-122° K 49/ it is almost certain that earth-designed cable materials will be too brittle and inflexible. More satisfactory materials may be developed that will insure sufficient flexibility and wear resistance, or (1)materials may be used that are suitable at lunar night temperatures but which must be coiled and stored in darkness during the lunar day or (2) earth-design materials may be used but with heating coils added so as to maintain the cable at a temperature at which it is not too brittle and inflexible. The latter solution would, of course, increase power consumption.

Centrally controlled rope-and-scraper systems would not require cables. Cables could be used with power shovel, dragline, scraper-loader, front-end loader, hoe or bulldozer systems 49 /. Multiple unit systems would require multiple cables and thus increase system weight and complexity. With cable power distribution, sufficiently large conductors could insure reasonably economic use of low voltage d-c as well as high voltage a-c power, but cable costs would be greater.

Although parts of the lunar surface "seen" in small scale by Luna IX and Surveyor I have rather gentle slopes 14 /, there are enough depressions and loose boulders that the surface may be quite abrasive to trailing cables. Until some "lunarock" is actually examined however, no estimate of its abrasiveness can be made.

5. Weight Relations: At the start of the project, it was thought possible that significant savings in equipment weight might be effected for postulated lunar designs over earth-designed models by replacing diesel and gasoline motors with electric motors. It is concluded that, while some weight savings may be made, the demands of the lunar environment may result in such reductions being small to moderate.

Motor frames are about the only place where substitution of light metals such as Be, Al or Ti can be made. The light metals generally do not have electrical properties suitable for substitution in the electrical parts of the motors or for shafts, bearings, etc.

The necessity for disposing of the heat generated by operating electric motors will add to their weight. Light fluids can be used and radiators, pumps and plumbing can be made of light metals but the requirement of integral cooling systems on all motors will add weight not required on earth.

Heat rejection systems may use liquid, vapor-liquid or gaseous working fluids. The liquid-vapor system has some advantages over the other two: however, the gas system is quite similar to liquid-vapor. It is simpler and may be multi-purpose, including lubricating bearings, cooling rotating machinery and actuating pressure devices 51/. Some advantages stated for liquid-vapor systems over liquid systems are (1) a smaller inventory of working fluid is needed because of the relatively large quantity of heat released by the fluid phase charges. (2) The vapor can be raised to a higher temperature than boiling. (3) The over-all heat coefficient is higher for liquid-vapor. (4) Not only is more fluid required by a liquid system but the heat rejector and plumbing to handle the fluid must be heavier. In comparison to a gaseous system, the vapor system is (1) lighter and requires less radiator area because of greater capacity per pound of fluid. (2) In a gas system, higher temperatures and higher pressures create more material and engineering problems 51/.

One disadvantage of the vapor system for space vehicles is that it has the problem of condensing in a nongravity condition. This condition will not exist on the lunar surface. The lunar gravity is much lower than on earth, but this should not result in operation any different from on earth.

The characteristics of the possible working fluids for either system will have an important bearing on any comparison of liquid-vapor and gaseous systems 51/.

Information collected on gasoline and diesel motors yielded a weight ranging from 535 to 2520 pounds for bulldozers and front-end loaders. For draglines and shovels, the range was from 4565 to 6620 pounds. There are undoubtedly models of similar capacities to these that will run a little heavier or a little lighter. The manufacturers of some of the heavier shovels and draglines considered, did not furnish motor weights.

Electric motors vary widely in weight with type and anticipated use. A representative of an electrical machinery manufacturer suggested a possibly attainable scale for a-c motors as follows 52/:

0.75-3.7 kw	(1-5 HP)	about 0.33 kg/kw	or 2.0 lb/HP	= 0.9-4.5 kg	(2-10 lb)
7.5-38.3 kw	(10-50 HP)	0.17 kg/kw	1.0 lb/HP	4.5-22.7 kg	(10-50 lb)
37.3-149.2 kw	(50-200 HP)	0.13 kg/kw	3/4 lb/HP	17.0-68.0 kg	(37.5-150 lb)
> 186.5 kw	(> 250 HP),	wt/HP ratio passes optimum and mechanical problems associated with physical size cause it to increase.			

Data furnished by another manufacturer 58/ gives, for 60 cycle induction motors, 230 volts at full load current, ~ 1800 rpm speed, drip-proof enclosures:

KW	HP	WEIGHT		WT/HP	
0.75	1	13.6 kg	(30 lbs)	18.1 kg/kw	(30 lb/HP)
3.73	5	31.8 kg	(70 lbs)	8.5 kg/kw	(14 lb/HP)
7.4	10	64.0 kg	(141 lbs)	8.6 kg/kw	(14.1 lb/HP)
37.3	50	181.4 kg	(400 lbs)	4.9 kg/kw	(8.0 lbs/HP)
74.6	100	367.4 kg	(810 lbs)	4.9 kg/kw	(8.1 lbs/HP)
111.9	150	542.1 kg	(1195 lbs)	4.8 kg/kw	(8.0 lbs/HP)

Standard DC motors, drip-proof enclosures, 240 volts DC:

KW	HP	WEIGHT		WT/HP	
0.75	1	31.8 kg	(70 lbs)	42.4 kg/kw	(70 lbs/HP)
3.73	5	63.5 kg	(140 lbs)	17.0 kg/kw	(28 lbs/HP)
7.46	10	147.4 kg	(325 lbs)	19.8 kg/kw	(32.5 lbs/HP)
37.3	50	394.6 kg	(870 lbs)	10.6 kg/kw	(17.4 lbs/HP)
74.6	100	598.8 kg	(1320 lbs)	8.0 kg/kw	(13.2 lbs/HP)
111.9	150	859.6 kg	(1895 lbs)	7.7 kg/kw	(12.6 lbs/HP)
186.5	250	1324.5 kg	(2920 lbs)	7.1 kg/kw	(11.7 lbs/HP)

There is a great contrast in the weight/HP ratios from the two manufacturers. Undoubtedly, the reason is that the information from the first manufacturer came largely from the aerospace electrical division and probably reflects experience in design of light motors for spacecraft and other weight-saving uses, thus representing the best yet achieved. Furthermore, the weights given may not include such things as drip-proof enclosures or totally enclosed motors, but refer to the bare working motor itself. The second manufacturer simply furnished data on earth-designed motors actually being used in industry. It is excellent that we have both sets of ratios. Rugged motors, designed for mining, probably cannot reach the most favorable ratios above, but the ratios in common industrial use on earth where weight is a factor, usually subordinate to efficiency, manufacturing cost and specific utility, should be surpassed.

It will be noted that for similar capacities, presently used industrial a-c motors have more favorable weight-horsepower ratios than do d-c motors. In general, for the same horsepower rating, d-c motors have ratios about 133 percent greater than those of a-c motors up to 37.3 kw (50 HP) and about 50 percent more, in the case of the larger motors.

It seems reasonable to postulate that a-c motors, designed for use in lunar mining machinery in the 7.5-112.0 kw (10-150 horsepower) range, might be designed to achieve a ratio of about 3.0 kg/kw (5 lbs per horsepower), not including the weight of heat rejection systems or of hermetically sealed bearing housings, etc. This would give weights of from 23-336 kg (50-750 lbs), significantly less than those of equally rated gasoline and diesel motors. Part of this savings will be offset by heat rejection systems, etc. Heat engines in use on earth have heat rejection or cooling systems but if they were to be used on the moon, these systems might have to be heavier and more elaborate, further adding to their weight.

Because of the probable small size of saving to be effected by change in motive power of earth equipment designs from heat engines, this factor has been omitted in the estimation of weight reduction factors for equipment designed for lunar use.

VI

FRAGMENTATION OF LUNAR ROCK WITH EXPLOSIVES

In all mining operations it is necessary to sever the ore and waste from the original mass. In unconsolidated material this can be accomplished by direct digging with the loading machine, but for coherent material, it is ordinarily accomplished by blasting. Depending on the conditions present either high explosives, expanding gas explosives or nuclear explosives may be used.

In this report, each method of rock blasting is analyzed, considering the lunar environment, and an estimate is made of the amount of material that would be required (supplies, etc.) from earth to produce a pound of water from lunar sources. The breakage of rock by explosives is first developed theoretically and then practical equations are derived for application to model lunar ore bodies.

1. Chemical High Explosives: It would be almost impossible to use presently available high explosives in the harsh lunar environment without elaborate handling equipment and storage facilities. However, based on the assumption that explosive manufacturers could, if required, produce high explosives that could be safely used in the lunar environment, the following application to lunar mining has been developed.

2. Theory of Rock Breakage: The investigation of the mechanism of failure resulting from the detonation of an explosive in a rock mass is complicated by the variety of the explosives available, the variety of materials blasted, the complex force system developed, and the short time of action involved. This complexity has resulted in the development of several theories of failure and innumerable empirically developed formulas for calculating charge sizes. To date, no theory has been developed that completely explains the blasting phenomenon.

The use of black powder as the first explosive naturally lead to development of the classical Gas Expansion Theory to explain the blasting phenomenon. This theory simply postulated that the failure of the rock was caused by the heaving force of the expanding gases produced by the rapid decomposition of the explosive.

The later development of high explosives, with their "shattering" effect, introduced the Shock Wave Theory to account for the rock breakage resulting from this type of explosive. The Shock Wave Theory considers the main failure process to be due to the scabbing effect caused by the reflection of the shock wave at free surface.

The intensified research during and since World War II, coupled with the development and use of many new "mixture" explosives and low velocity high explosives, has resulted in the development of several theories which consider the dual action of the shock wave and gas pressure, as well as the resistance of the material being blasted, in explaining the mechanism of rock failure.

The mechanism of rock breakage by blasting can best be understood by tracing the sequence of events as they occur beginning with the initiation of the explosive.

3. Mechanism of Detonation: The process of detonation is the factor that differentiates high explosives from low explosives which only burn at a rapid rate. According to Loving (1964), the definitive characteristic of the chemical reaction in a detonation is that it is initiated by, and in turn supports, a supersonic shock wave proceeding through the explosive. The chemical reactions which support and control the stability and velocity of the detonation front are those which occur in the primary reaction zone (Fig. 15). This zone is a region of rapid, but not instantaneous, reaction located immediately behind the detonation front. It is bounded at the rear by the Chapman-Jauquet plane, which separates it from the region of relatively slower secondary reactions. Secondary reactions, which are significant in mixed explosives, can affect the final performance of the explosive, but they have no effect upon the stability or velocity of the detonation front.

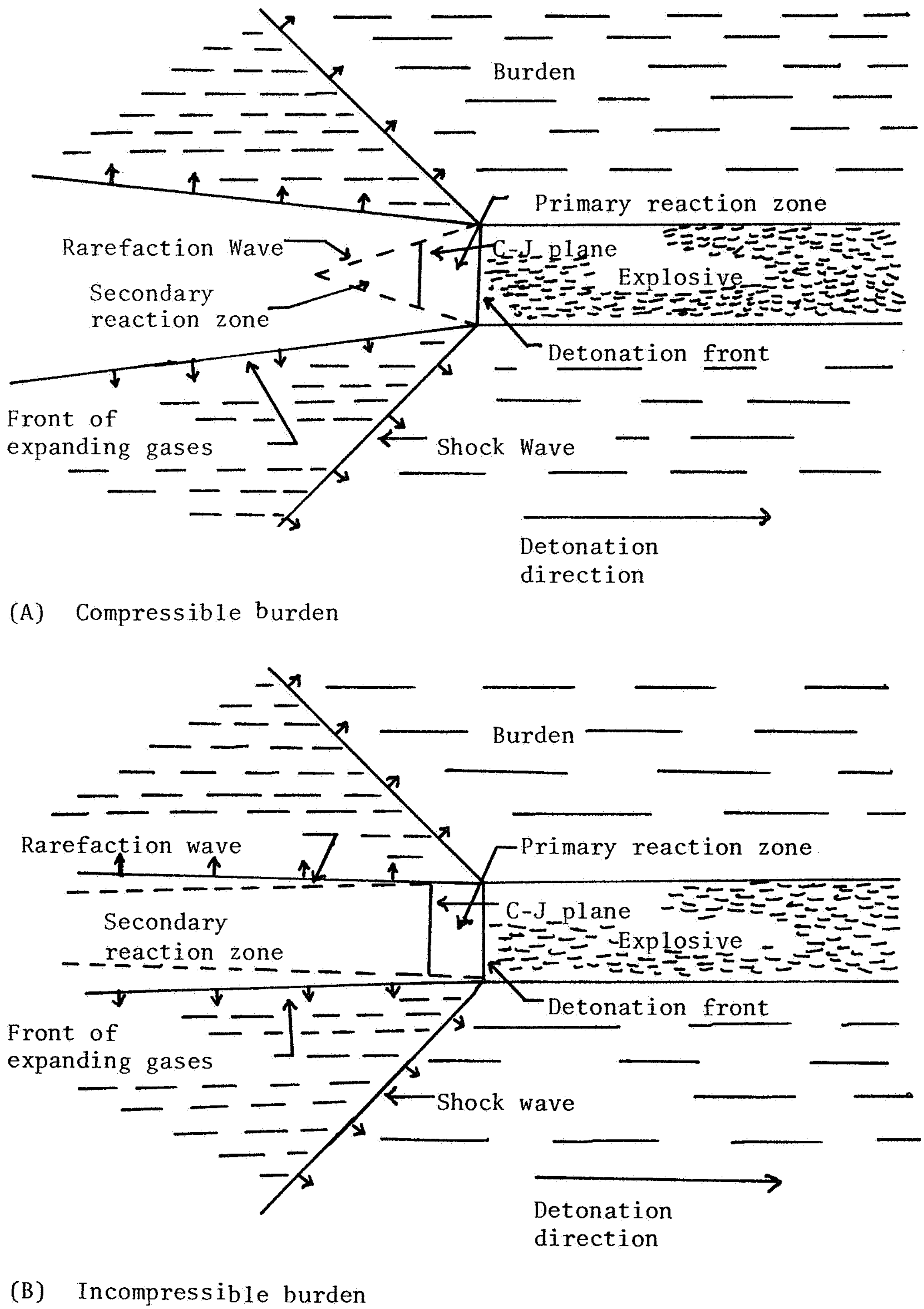


Fig. 15. Diagram of detonation mechanism in column charge (after Loving 1964).

Also of importance in supporting the detonation is the nature of the confining burden 59/. If the burden is compressible (air, water, or soft rock), there will be a rapid loss of energy which will result in a lower detonation temperature and pressure. This lost energy is expended by the rapid expansion of the gaseous products in compressing the confining burden. These energy losses produce a rarefaction wave in the reaction zone which removes support from the detonation front. The result is a slower than ideal detonation velocity. If the energy losses are great enough, a complete detonation failure will occur (see Fig. 15A).

In an incompressible burden (hard rock) very little energy is lost because the gaseous products cannot expand (see Fig. 15B). This allows for a maximum reaction time at the peak detonation velocity, temperature, and pressure. This in turn allows the relatively slower secondary reactions time to supply more useful energy to the blast force before the burden begins to break.

The peak detonation pressures for high explosives range from 20,000 to 200,000 atmospheres, and the temperatures of reaction range from 1,700°K to 5,800°K 60/. The detonation velocities, which are given by manufacturers, range between 9,000 and 30,000 fps.

4. Generation of the Shock Wave: The main factors which control the formation of a shock wave are the velocity of the detonation front and the acoustical velocity of the confining burden. If the velocity of the detonation front is supersonic with respect to the acoustical velocity of the burden, a shock wave is formed which travels through the rock at a velocity ranging from 10,000 to 16,000 fps 61/. The shock wave dissipates energy rapidly causing a rapid decline in the temperature and pressure of the blast. Most blasting experts agree that the energy contained by the shock wave is approximately 9 percent of the total explosive energy generated. 61/.

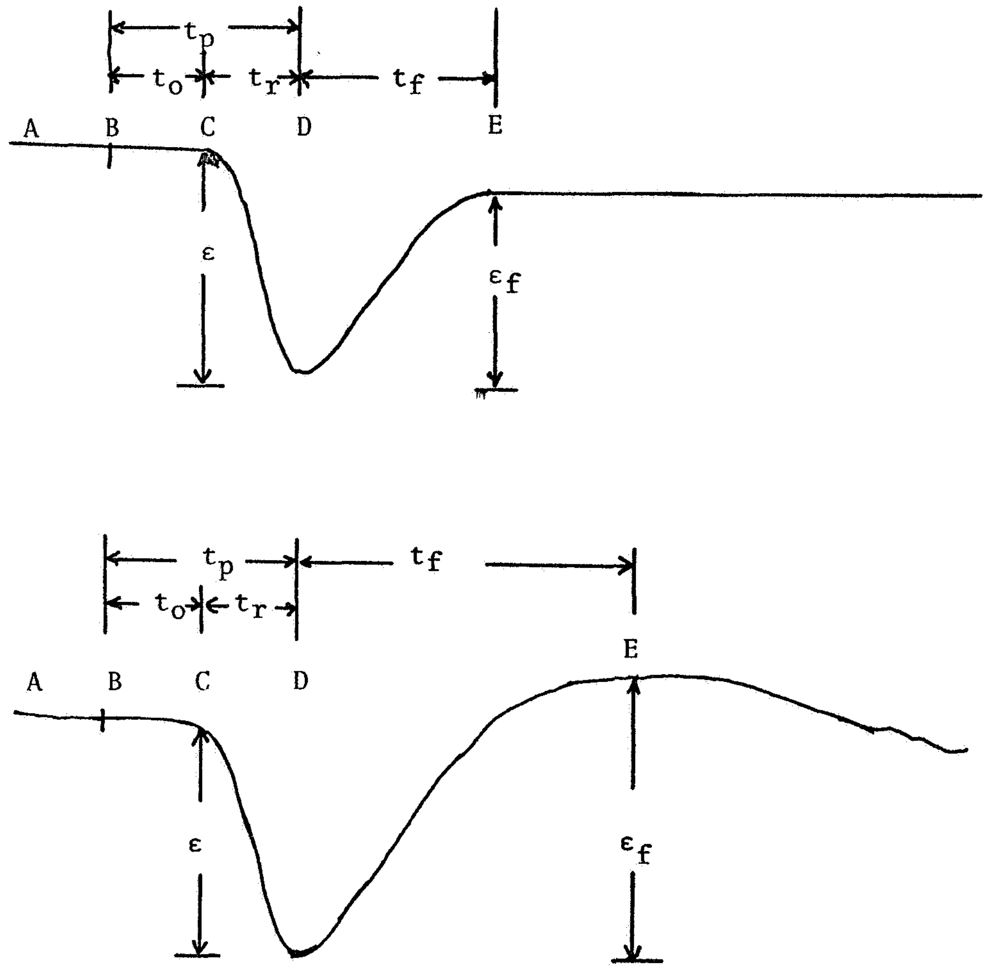
If the detonation velocity is subsonic relative to the acoustical velocity of the confining burden, no shock wave is produced. The drop in temperature and pressure is less rapid and the immediate loss of explosive energy is small and occurs slowly as a pressure transient traveling at sonic velocity through the rock.

Both of these cases occur in rock blasting, but a shock wave is usually produced. In order to explain the effects of the shock wave upon the rock, it is necessary to first look at the properties of stress waves in an elastic solid.

5. Properties of Stress Waves: From Kolsky 62 / we learn that an unbounded elastic medium (rocks are considered to behave elastically in most cases) is capable of transmitting two types of stress waves at different velocities. The faster of the two is called the dilatational or longitudinal stress wave because the particle movement is in a direction parallel to the direction of wave propagation. The slower stress wave is called the distortional or tangential stress wave because the movement of the particles is in a direction perpendicular to the direction of wave propagation.

The longitudinal stress wave can exist either as a compressive pulse or a tensile pulse. The longitudinal stress wave as it exists in the shock wave is characterized by a single compression of short duration having a rapid rise time and slower decay time 63 /. The predominant parts of a normal compressive shock pulse are shown in the trace in Fig. 16. An important property of the longitudinal stress wave is that if it impinges on a free surface as a compressive pulse, it will be reflected as a tensile pulse of almost equal magnitude. The opposite occurs if the impinging pulse is tensile. This process occurs because the density of the air is much less than that of the rock, and the resultant stress at the free face must be zero. Also of interest are calculations which show that the original compressive longitudinal stress wave contains tensile stresses at a great distance from the borehole 61 /. This change in stress is shown in Fig. 17B, but is thought to be of little practical effect in explaining rock breakage.

The shock pulse as generated by a detonating explosive also creates a lateral pressure. The variations of the state of stress that exist in the lateral direction are



A = Start of trace

B = Detonation of charge

C = Start of strain pulse

D = Peak of compressive pulse

E = End of fall strain

t_o = Arrival time for start of pulse

t_r = Rise time

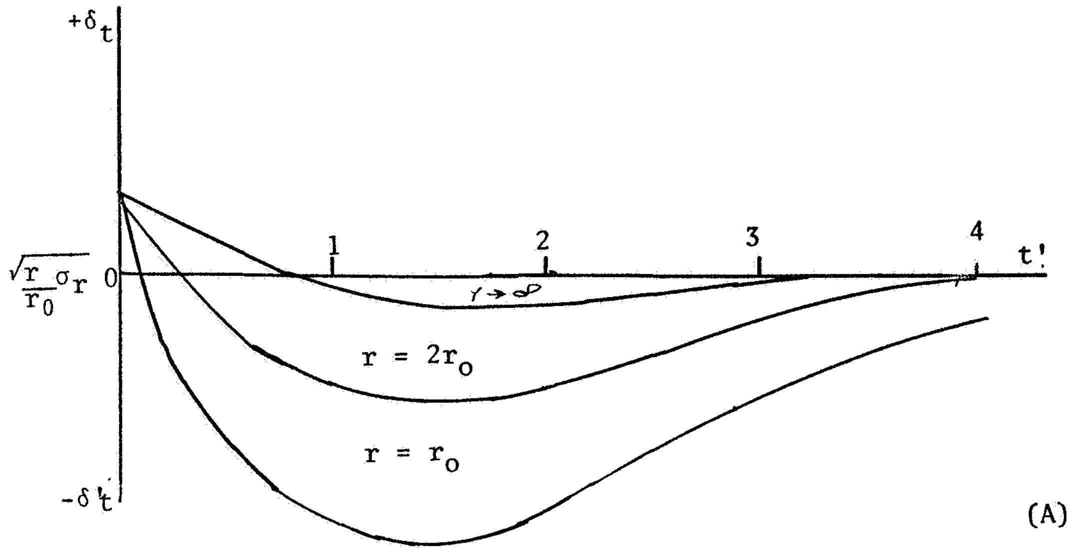
t_f = Fall time

t_p = Arrival time for peak strain

ε = Peak compressive strain

ε_f = Fall strain

Fig. 16. Trace of strain pulse (reprint from Bureau of Mines RI-5356).



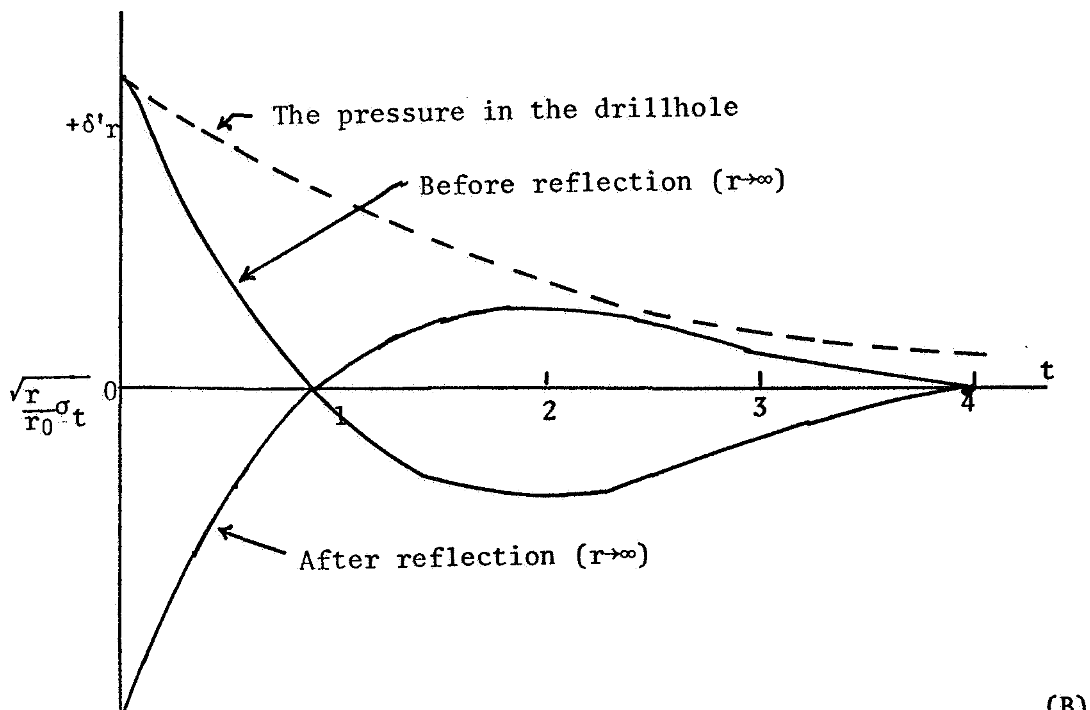


Fig. 17. The stress in a shock wave from a cylindrical hole with radius r_o of the drillhole. (A) the tangential stress of the wave before reflection (B) the radial stress of the wave (after Langefors and Kihlstrom 1963).

shown in Figure 17A 61/. As the shock pulse arrives at a given point, the lateral stress is compressive, but it rapidly falls to negative values as the pulse passes giving rise to tensile stresses. In the region near the borehole this lateral tensile stress will exceed the compressive stress of the shock pulse.

The failure of solids subjected to stress pulses is controlled mainly by the following three characteristics of stress waves 62/.

(1) The stress pulse affects only a small portion of the material at a given instant of time. This allows the failure in one region to occur independent of the state of stress in another region.

(2) The velocity of crack propagation is generally considerably slower than the velocity of the pulse. For short pulses the crack may not have time to form before the pulse has passed and the stress removed.

(3) The interference of the stress waves can cause superposition of the stresses and create forces greater than any of the original pulses, thereby, bringing about failure that would not have occurred otherwise.

Energy losses by stress pulses, aside from that used in breaking the rock, occur as spherical and frictional attenuations 64/. Spherical attentuation decreases the stress of the wave as it radiates outward from the borehole. This decrease in stress is inversely proportional to the distance of the pulse from the source.

The frictional attenuation depends on the natural resistance of the material to rapid variations in the stress level. It is a form of stress hysteresis in which the energy is removed from the wave and transformed into heat.

These losses of energy reduce the effects of the stress pulses through the rock from the detonating explosive.

6. Rock Breakage by the Shock Wave: The rock failures resulting from the action of stresses created by the shock wave can be broken into three main types: (1) the crushing action near the borehole by the radial compressive stress, (2) the

radial cracks formed by the lateral tensile stress, and (3) the scabbing action caused by the reflection of the shock wave at the free surface.

In the region immediately surrounding the borehole, the radial compressive stress of the shock pulse is of sufficient strength to overcome the compressive strength of the rock, and a zone of superfine crushing is produced 63 /. The thickness of this shattered zone is usually approximately equal to the radius of the borehole 61/. The energy lost by the shock pulse in crushing the rock results in the peak stress of the pulse being reduced to the compressive strength of the rock 65/. However, the impact action of the shock pulse increases the compressive strength of the rock immediately surrounding the charge. This results in less than expected crushing, and a higher peak stress for the pulse. Therefore, the peak compressive stress of the shock pulse that travels out into the rest of the rock mass is controlled by the compressive strength of the rock in the zone of crushing under the conditions present during the blast.

In the region near the borehole, the lateral tensile stress created by the shock pulse is of sufficient strength to cause the rock to fail in tension 61 /. The result of this failure is the development of a system of primary radial cracks around the borehole. The direction of formation of these cracks greatly influences the final failure process.

During these first two stages of failure, which occur in less than one millisecond 61/, there is practically no useful fragmentation. Unless a free surface is near the borehole, the extent of damage will be the partial comauflet produced by the crushing action of the shock pulse. However, if a free surface is present near the borehole, there will be additional failure.

As the compressive shock pulse impinges on the free surface, it is reflected as a tensile pulse that can cause scabbing of the surface rock if the strength of the pulse is great enough 61/. The scabbing process can best be explained by assuming the impinging pulse to be triangular shaped as shown in Fig. 18A 63/. As the resultant sum of the reflected tensile pulse and the remainder of the original pulse exceeds the tensile strength of the rock, a crack is formed (see Fig. 18B) and a slab is broken off and moves forward forming a new free surface (Fig. 18C). The remainder of the original compressive pulse is reflected at this new surface and the process of breakage is repeated forming another slab (Fig. 18D). The scabbing process continues until the energy of the original compressive pulse is consumed. The energy trapped in each slab is consumed in throwing the slab forward. Hino 66 / gives the following theoretically developed equations for calculating the number of slabs that will be formed and the length of each slab:

$$\text{Number of slabs} \quad N = \frac{L}{2T} \quad \text{or} \quad \frac{Pa}{St} \tag{1}$$

$$\text{Thickness of slab} \quad T = \frac{L}{2} \cdot \frac{St}{Pa} \tag{2}$$

where L is length of pulse,
T is thickness of slab,
Pa is peak pressure of pulse, and
St is tensile strength of rock.

The results of many experiments conducted by Hino, Duvall, Atchison, and others have proved that scabbing can occur. However, according to Langefors and Kihlstrom 61 /, in order for this process to achieve importance as a mechanism of rock failure the specific consumption of explosives must considerably exceed 1 Kg/m^3. This suggests that some other process must be responsible for the final breakage of the rock.

7. Rock Breakage by Gas Expansion: The first stages of rock breakage in so far as they occur are affected by the shock wave. However, when blasting with low explosives the shock wave as such is not formed. Therefore, the breakage is caused by the widening of existing cracks close to the borehole by the force of the expanding

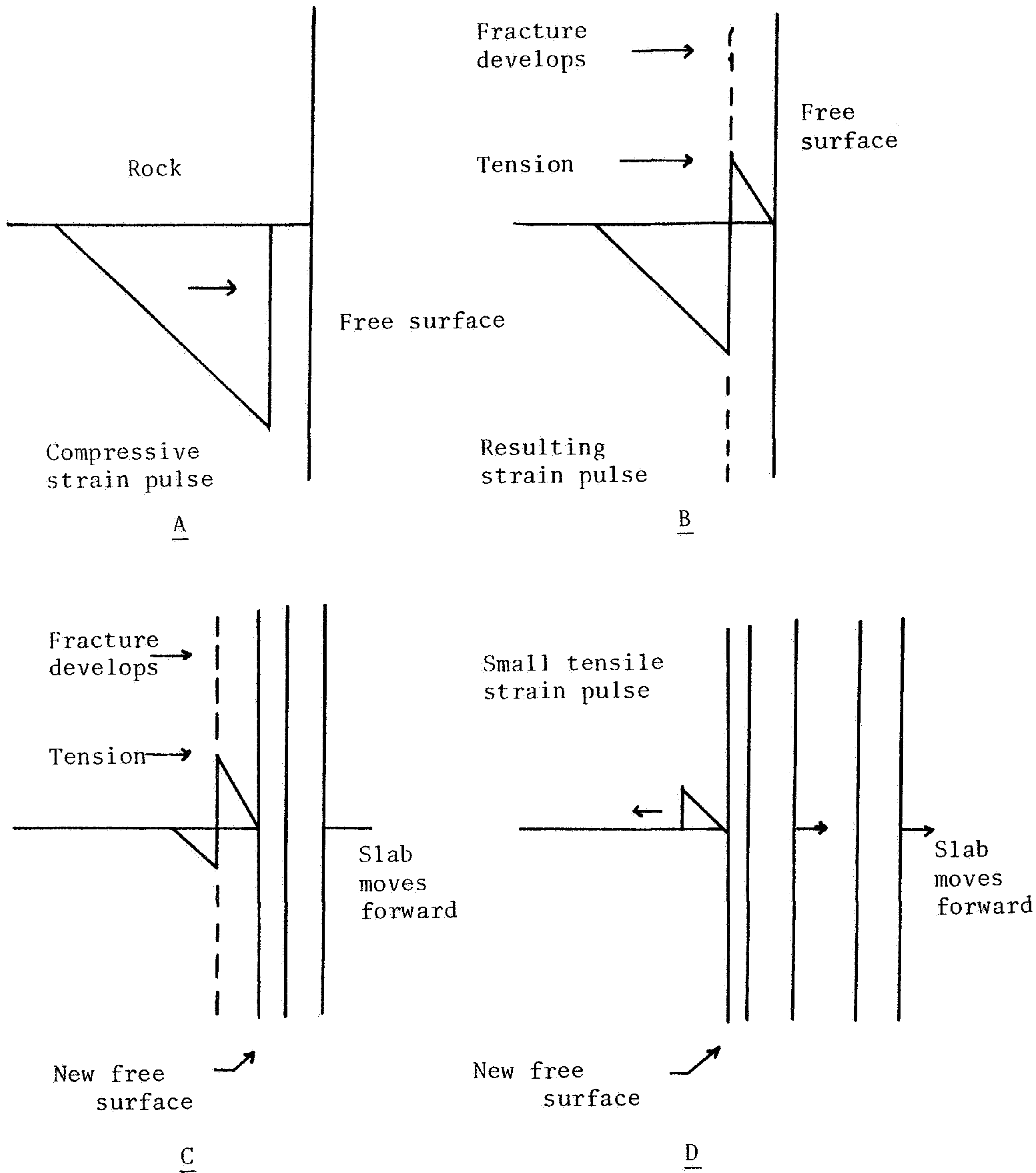


Fig. 18. Tensile fracture by reflection of a compressive strain pulse (reprinted from Bureau of Mines RI-5356).

gases 61 /. Since these cracks usually have no favorable direction, the burden in most cases is not completely loosened. Langefors and Kihlstrom 61/ have reported the following observations which are of interest in this discussion.

If two visually insignificant indications for cracks are chiselled to give an apparent 90° breakage angle, the breaking force of black powder, in relation to the gas volume and energy content, appears to be the same as for high explosives. This is very illustrative because it shows that the energy necessary to completely loosen a given burden has no correlation with the shock wave. The conclusion then is that the shock wave is not responsible for the actual breakage in rock blasting but only provides the basic conditions for breakage.

The last stage of breakage is a much slower process than the preceding stages. Under the influence of the high gas pressure generated by the explosion, the primary radial cracks previously formed and any naturally existing flaws are expanded. As the cracks expand, the free surface yields and moves forward. This action can be described as a semistationary process in which the stress pattern at any moment decides the continuation of the cracks 61 /. As the free surface moves forward, the gas pressure is reduced, but the tension in the cracks is increased because of the outward bending of the burden. If the burden is not too great, some of the cracks will extend to the surface and the rock mass will be completely loosened and torn off.

8. Shape of Blast Force Propagation: The shape and direction of the blast force propagation are very important in determining the amount of rock that will be broken and the direction it will be thrown in a given blast. According to Kochanowsky 67/, the design of the charge and the method of detonation are of immediate influence in determining the direction and shape of the force propagation. The shape of the shock wave front depends on the relationship between the detonation velocity of the explosive and the velocity of the shock wave in the burden. The shape of the gas pressure front is determined by the effects of the stemming and the resistance of the

rock to breakage. The general shape of the explosive force propagation in a borehole blast is shown in Fig. 19. Of interest are measurements which show that the peak pressure of the shock wave is greater in the portions relatively parallel to the borehole than in the portions radiating from either end 68 /.

The shape of the explosive force propagation will not affect the basic mechanism of breakage. However, if the shape of the blast force propagation can be predicted in advance or possibly controlled, the blast energy can be concentrated and guided in a desirable direction. This action can increase the efficiency of a blast by reducing the consumption of explosives and increasing the resulting fragmentation. Knowledge of the factors influencing the shape of the blast force propagation can be very useful in planning blasts.

9. Calculation of Explosive Charge: The determination of the quantity of explosive, the blasting pattern, and the detonation sequence for effective rock breakage are the main problems that must be solved for efficient blasting of any rock mass. The trial and improvement method of developing satisfactory blasting procedures is expensive but feasible in earth operations; however, it would be essentially impossible for lunar operations. In addition, there is no way of determining if a method so developed is the most efficient or economical.

The present state of knowledge concerning the mathematical calculation of the explosive charge has been developed to a fairly high degree, but much remains to be done in applying it to practical operations. Most formulas to date have been based on a particular usage, with no general formula available. This section of the report will attempt to trace the development of current formulas and modify then for lunar use.

10. General Case - Crater Blasting: The force generated by an explosive must break as well as move the rock; it has to overcome the resistance against

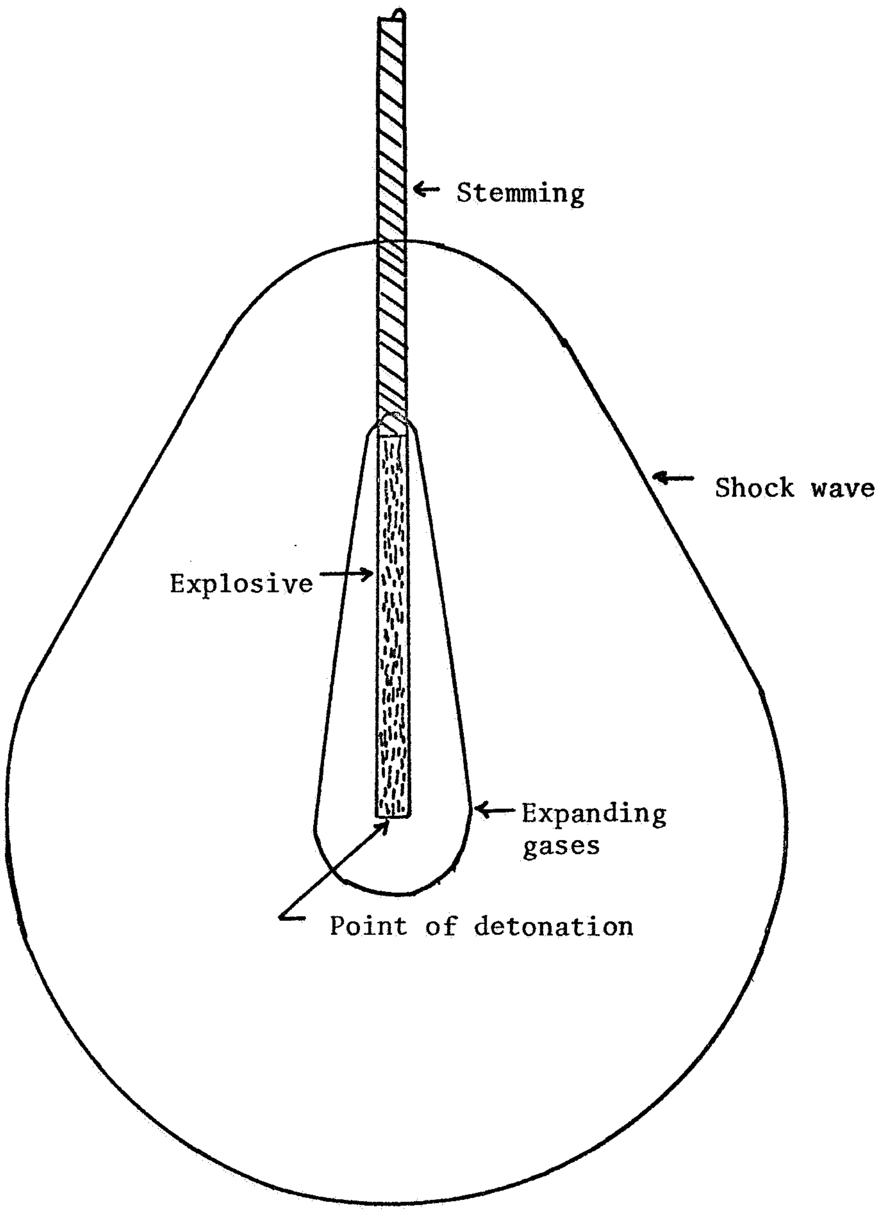


Fig. 19. Shape of force propagation from borehole blast.

gravitational forces 69 /. Belidor recognized this dual resistance of rock as early as 1725. He stated that one part of the change was proportional to the strength of the rock and the other proportional to the volume excavated. He proposed the formula

$$Q = aV^2 + bV^3 \tag{3}$$

where Q = explosive charge
V = burden (least line of resistance)
a = constant based on rock strength
b = constant based on gravitational force

Because early interest centered about military applications (i.e., the formation of craters in weathered surface rock), Belidor's work was quickly forgotten and replaced by the classical cube-rule (proposed earlier by Vauban) which is based on the law of conformity. Simply stated, the explosive charge is proportional to the volume of the crater produced. The cube-rule is expressed as:

$$Q = kv^3 \tag{4}$$

where k = constant based on rock type.

Based on the results of extensive research since World War II, Langefors and Kilhstrom 61 / have developed the following general formula which is an extension of Belidor's formula to include a "throw" or "swell" component:

$$Q = k_2 V^2 + k_3 V^3 + k_4 V^4 \tag{5}$$

where k_2, k_3, and k_4 are constants dependent on rock type. They additionally reported that for ordinary bed rock:

$$Q = .01 V^2 + .40 V^3 + .004 V^4 \tag{6}$$

where Q is in kg and V is in meters.

A close look at formula (5) shows the following facts of interest:

(1) For small blasts ($V < 1$ meter) the strength of the rock is of most importance.

(2) For relatively large high explosive blasts ($1 < V < 10$ meters) the volume of material to be excavated is of greatest importance.

(3) For larger high explosive or nuclear blasts ($V > 10$ meters) the amount of throw is of considerable importance.

Kochanowsky 70/, based on the results of numerous tests blasts, had also suggested the variability of rock resistance as related to the size of the blast. He postulated that:

(1) The larger the piece of rock to be blasted, the greater the points of weakness and, therefore, the smaller the specific strength.

(2) The larger the burden, the further the material must be thrown for effective excavation.

These considerations explain the past failures resulting from the use of the cube-rule to predict the results of large blasts from the analyses of small test blasts.

The equation developed by Langefors and Kilhstrom 61/ is the general prediction formula for crater blasts, and as such gives reliable results when applied with the appropriate constants for each rock type.

Assuming ordinary lunar rock to be similar in strength and mass to earth bed rock, the prediction equation for crater blasts can be applied to determine if high explosives could be used economically for blasting lunar rock.

Based on the preceding assumption, the first term of equation (6) would remain the same, but the second and third terms would have to be modified to account for the change in gravity. Since lunar gravity is approximately 1/6 that of earth gravity, the second term would be $.07\ V^3$ and the third $.0007\ V^4$. This indicates a greater relative quantity of the explosive will be required to break the rock in lunar operation than on earth. Rewriting the equation we have:

$$Q_L = .01\ V^2 + .07\ V^3 + .0007\ V^4 \tag{7}$$

where Q_L is lunar explosive charge in kg.

A more meaningful comparison can be made if the equations are converted to calculate the specific charge (kg/m^3). This can be accomplished by dividing through by V^3 to obtain:

$$q = \frac{.01}{V} + .4 + .004V \tag{6a}$$

$$q_1 = \frac{.01}{V} + .07 + .0007\,V \tag{7a}$$

where q is specific charge on earth (kg/m^3)

q_1 is specific charge on moon (kg/m^3)

Calculation of the specific charge required for burdens ranging from .01 meters to 100 meters have been calculated and shown in Table 9 for comparison. Also included is a ratio of the lunar specific charge to the earth specific charge (n).

Table 9

Specific Charge Calculations for
Effective Fragmentation on the Earth and Moon

Burden (Meters) V	Specific Charge (kg/m^3) Earth (q)	Moon (q_1)	Ratio q_1/q (n)
0.01	1.400	1.070	0.764
0.10	0.500	0.170	0.340
1.00	0.414	0.081	0.196
10.00	0.441	0.078	0.177
100.00	0.800	0.140	0.175

The calculations shown in Table 9 indicate that the most efficient blasts are those conducted with burdens between 1 and 10 meters on both the earth and moon. Also it will be noted that as the burden increases, the ratio n decreases and approaches 1/6.

It is a well known fact that in earth operations, the specific charge for surface mining blasts varies from approximately .2 kg/m^3 for soft rock to .6 kg/m^3 for hard rock. Assuming the same relationship for production blasting on the earth and moon as for crater blasting, it can be predicted that the specific charge for lunar bench blasting will vary from approximately .04 kg/m^3 for soft material to .12 kg/m^3 for hard material.

Assuming that most of the "ores" to be mined initially will have a mass of approximately 2,454 kg per cubic meter, it will require between 16.5 gm/metric ton (0.033 lbs/ton) mass to 49 gm/metric ton (0.098 lbs/ton) mass of explosive to effectively fragment and excavate the ore.

Because of a lack of a definite cost figure for sending supplies to the moon, Table 10 has been developed to give an idea of the quantity of explosive in kg required to produce one kg of water for various grades of "ore."

Table 10

Estimated Quantity of Explosive
Required to Produce Water on the Moon

Grade of Ore (%)	Explosive required (kg per kg water) Soft "Ore"	Hard "Ore"
1%	.0017	.0049
5%	.0003	.0010
10%	.0002	.0005
100% (ice)	.00002	-----

From Table 10 it can be concluded that the relative cost of explosives necessary to produce water on the moon is small, compared to the cost of shipping water to the moon assuming a constant cost per pound for supplies.

11. Bench Blasting: The most widely used production blasting pattern in surface mining is the bench blast. The design of the bench pattern and the calculation of the explosive charge necessary for effective rock breakage is a function of the rock type, drill hole diameter (d), bench height (K), burden (V), spacing (E), and explosive type (see Fig. 20 for clarification of terminology).

It is necessary in most cases to divide the charge calculation into two parts: (a) a concentrated bottom charge for effective breakage of the toe, and (b) an evenly distributed column charge for efficient fragmentation of the rock mass.

A. Calculation of bottom charge: The concentrated bottom charge can be calculated by a formula similar to the crater formula (see equation 6) 74 /. It can be written as

$$Q_0 = a_2 V^2 + a_3 V^3 + a_4 V^4 \qquad (8)$$

Where Q_0 is the bottom charge (Kg), and
a_i is a special case of K_i for
$K/V = 1$, and the explosive charge concentrated at the bottom of the hole ($h/V \approx O$, see Fig. 20).

Test blasts conducted by Langefors and Khilstrom 61 / have produced values of $a_2 = .07$ kg/m^3, $a_3 = .40$ Kg/m^3, and $a_4 = .004$ Kg/m^3 for typical bed rock. Substituting these values into equation 8, gives us

$$Q_0 = .07\ V^2 + .4\ V^3 + .004\ V^4 \qquad (9)$$

for calculating the bottom charge. In practice it is impossible to concentrate the bottom charge ($h/V \approx O$); therefore the drill hole is drilled a depth of .3V below the toe of the bench. The bottom charge is then placed in the bottom 1.3V length of the hole and produces the results of a concentrated charge for practical purposes (see Fig. 21).

B. Calculation of the column charge: The column charge is placed in the remainder of the hole as shown in Fig. 21. It can be calculated by an equation of the form

$$Q_c = b_2 V^2 + b_3 V^3 + b_4 V^4 \qquad (10)$$

where Q_c is column charge (kg), and
b_i is a special case of K_i for
$K \gg V$ and $h > V$.

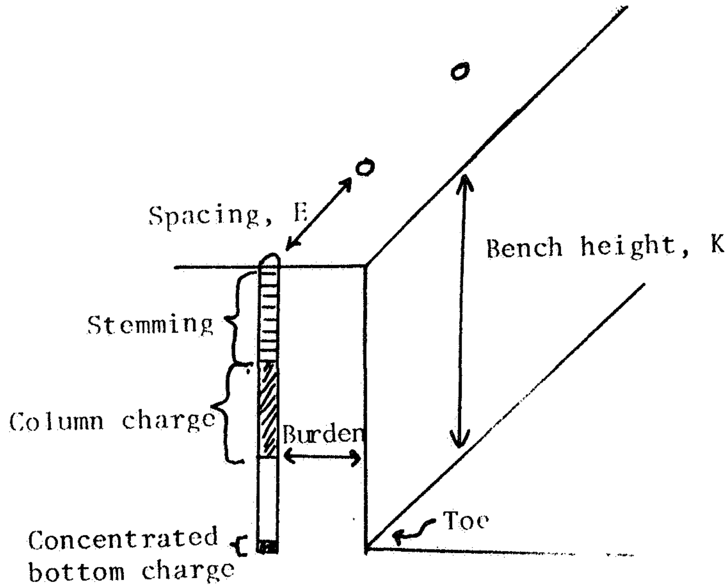


Fig. 20. Bench blast pattern with concentrated bottom charge.

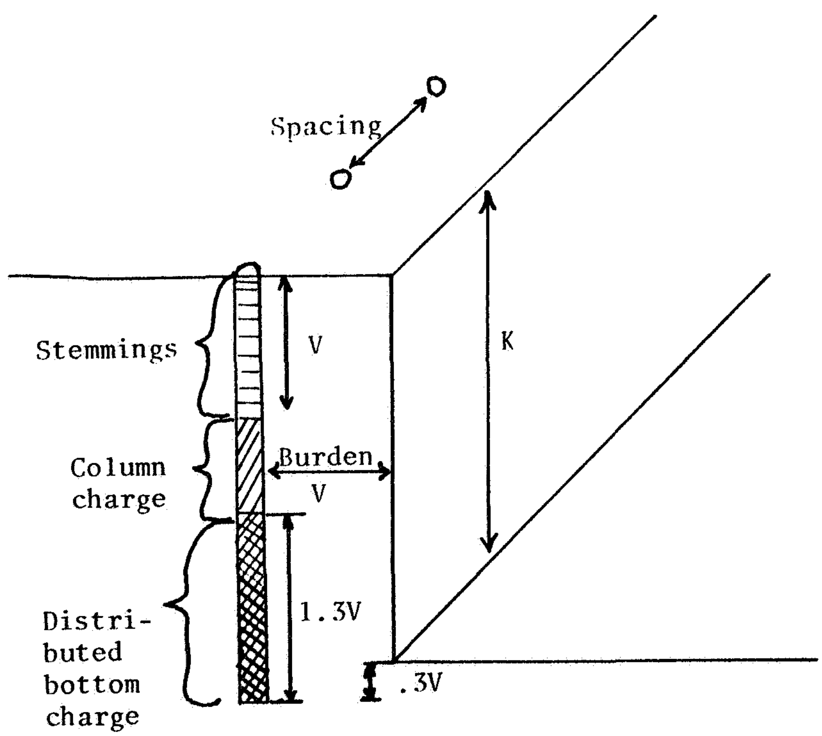


Fig. 21. Typical bench blast pattern.

To be useful (because of variation in bench height, K) equation (10) must be divided through by V to yield a charge per meter of drill hole, or:

$$1_c = b_2V + b_3V^2 + b_4V^3 \qquad (11)$$

where 1_c is charge per meter of drill hole (Kg/m).

Experimental blasts conducted by Langefor and Khilstrom 61/ have shown the b_i values to be related to the a_i values for most rock types as:

$$\left.\begin{array}{l} b_2 = .4a_2 \\ b_3 = .4a_3 \\ b_4 = \text{negl.} \end{array}\right\} \qquad (12)$$

The total charge (Q_i) can now be determined by multiplying equation (11) by the column length of the hole (K-V) and adding to it the concentrated bottom charge Q_0. This produces:

$$Q_t = Q_0 + 1c(K-V) \qquad (13)$$

By substituting the values of equation (12) and simplifying, the equation can be written as:

$$Q_t = .4a_2(K/V + 1.5)V^2 + .4a_3(K/V + 1.5)V^3 + a_4V^4 + b_4(K/V - 1)V^4. \quad (14)$$

This formula can also be written as a form of equation (6) as:

$$Q_t = K_2V^2 + K_3V^3 + K_4V^4 \qquad (15)$$

where Q_t = total charge per drill hole (kg/m^3)
$K_2 = .4a_2(K/V + 1.5)$
$K_3 = .4a_3(K/V + 1.5)$, and
$K_4 = a_4$, b_4 is negl.

Substituting the values of a_i for typical bed rock:

$a_2 = .07$ kg/m^3
$a_3 = .4$ kg/m^3
$a_4 = .004$ kg/m^3

we get for the total charge per drill for ordinary rock:

$$Q_t = k_2V^2 + k_3V^3 + k_4V^4 \qquad (16)$$

where $k_2 = .028\ (K/V + 1.5)$
$k_3 = .16\ (K/V + 1.5)$
$k_4 = .004$

This equation is valid only for cases of $K/V > 2$. For values of $K/V < 2$, the cases of the concentrated charge and the column charge overlap. This gives a higher specific charge and is therefore generally not used in production applications.

Equation (16) can be revised for lunar applications by considering the effects of the gravity variations on the terms k_3 and k_4 as previously done in the case of crater blasts. This produces a modified version of equation (16) in the form:

$$Q_1 = d_2V^2 + d_3V^3 + d_4V^4 \qquad (17)$$

where Q_1 = total lunar charge per drill hole, kg
$d_2 = K_2 = .028\ (K/V + 1.5)$
$d_3 = K_3/6 = .027\ (K/V + 1.5)$
$d_4 = K_4/6 = .0007$

Equations (16) and (17) have been used to calculate values of the total charge per drill hole for various values of V and K as shown in Table 11. Also for comparison, the specific charge (kg/M^3) has been determined by dividing the total charge (kg) by the volume of material broken (V^2K). In addition, a ratio (n) of the lunar charge (q_1) to the earth charge (q) has been determined.

From Table 11, it can be seen that the explosive consumption per unit volume of "ore" increases with an increase in the burden (V). Also with an increase in K/V if V is held constant. This leads to the general conclusion that the next economical blasting can be accomplished with a relatively high bench and small burden.

However, the cost per unit volume of "ore" for drilling the blast holes will vary inversely with the square of the burden assuming a constant cost per unit length of drill hole. It will be necessary therefore to consider the drilling cost plus the explosive cost to arrive at an economic blast pattern.

Because of the unknown nature of mining cost in the lunar environment, the "costs" of drilling and explosives have been related to the amount of water produced. Assuming a 1% grade ore, the relative blasting cost has been determined by calculating the unit weight of explosive required for blasting. These values are shown in Table 12 and were derived from approximations taken from Table 11. It is also

Table 11

Explosive Consumption for Various Bench Patterns

Burden V (m)	Height K/V	Volume V^2K	Earth Q_t	Earth q_t	Lunar Q_1	Lunar q_1	q_1/q_t
0.5	2	0.5	0.095	0.19	0.04	.07	.38
0.5	4	1.0	0.149	0.15	0.06	.06	.38
0.5	10	2.5	0.311	0.12	0.12	.05	.39
1.0	2	2.0	0.662	0.31	0.19	.09	.28
1.0	4	4.0	1,040	0.26	0.30	.08	.29
1.0	10	10.0	2.170	0.22	0.63	.06	.29
2.0	2	8.0	4.940	0.62	1.16	.14	.23
2.0	4	16.0	7.720	0.48	1.82	.11	.24
2.0	10	40.0	16.100	0.40	3.78	.10	.24
5.0	2	50.0	74.900	1.50	14.70	.29	.20
5.0	4	100.0	116.000	1.16	22.80	.23	.20
5.0	10	250.0	241.000	0.96	47.30	.19	.20
10.0	2	200.0	609.000	3.05	115.00	.56	.18
10.0	4	400.0	935.000	2.34	174.00	.44	.18
10.0	10	1,000.0	1,912.000	1.91	353.00	.35	.18

Table 12

Relative Drilling and Explosive Cost
(Lbs of Explosive Per Lb of Water Produced)

Burden V (m)	Explosive Cost (lbs/lb water) \$E	Drilling Cost (\$D)		
		\$D = \$E	\$D = 10\$E	\$D = 100\$E
0.5	.003	.0160	.160	1.600
1.0	.004	.0040	.040	0.400
2.0	.005	.0010	.010	0.100
5.0	.010	.0001	.016	0.016
10.0	.020	-	negl.	0.004

assumed that the unit drilling cost for a burden of V = 1 will be equal to or greater than the explosive cost per unit volume. As shown in Table 12, three cases have been calculated for the drilling cost vs various burdens (V). These are: (1) the drill cost per unit volume equal to the explosive cost per unit volume, (2) the drilling cost equal ten times the explosive cost per unit volume, and (3) the drilling cost equal one hundred times the explosive cost per unit volume.

Plots of the relative drilling and explosive costs are shown in Fig. 22 and the total cost curve obtained. It is evident from this plot that the determination of the most economic pattern (burden) will be greatly dependent on the cost of drilling. A good estimate from the curve would be that a blast pattern with V in the range of 1 to 10 would be the most economic. However, if drilling costs are higher than the above assumed estimates, the burden will have to be made larger.

It is evident that for a low grade "ore" (1% water) the fragmentation cost for explosives and drilling will be less than .1 the cost of the value of lunar water. A higher grade "ore" would reduce this cost considerably.

12. Expanding Gas Explosives: Three basic types of expanding gas explosives are presently being used in coal mine operations 71 /. They are the Airdox, Cardox, and Hydrox Systems. Basically, each type releases a gas under a pressure of 10,000 - 30,000 psi within the borehole. Normal burdens are .5 meters and the "explosive" consumption approximately .2kg/m^3 of coal produced.

At present it has not been demonstrated that such explosives can be used to fragment materials (rock) stronger than coal. For this reason, the authors feel this type of explosive would not be available for lunar operations. Also the "explosive" consumption is practically the same as for high explosives, which eliminates any possibility of supply savings.

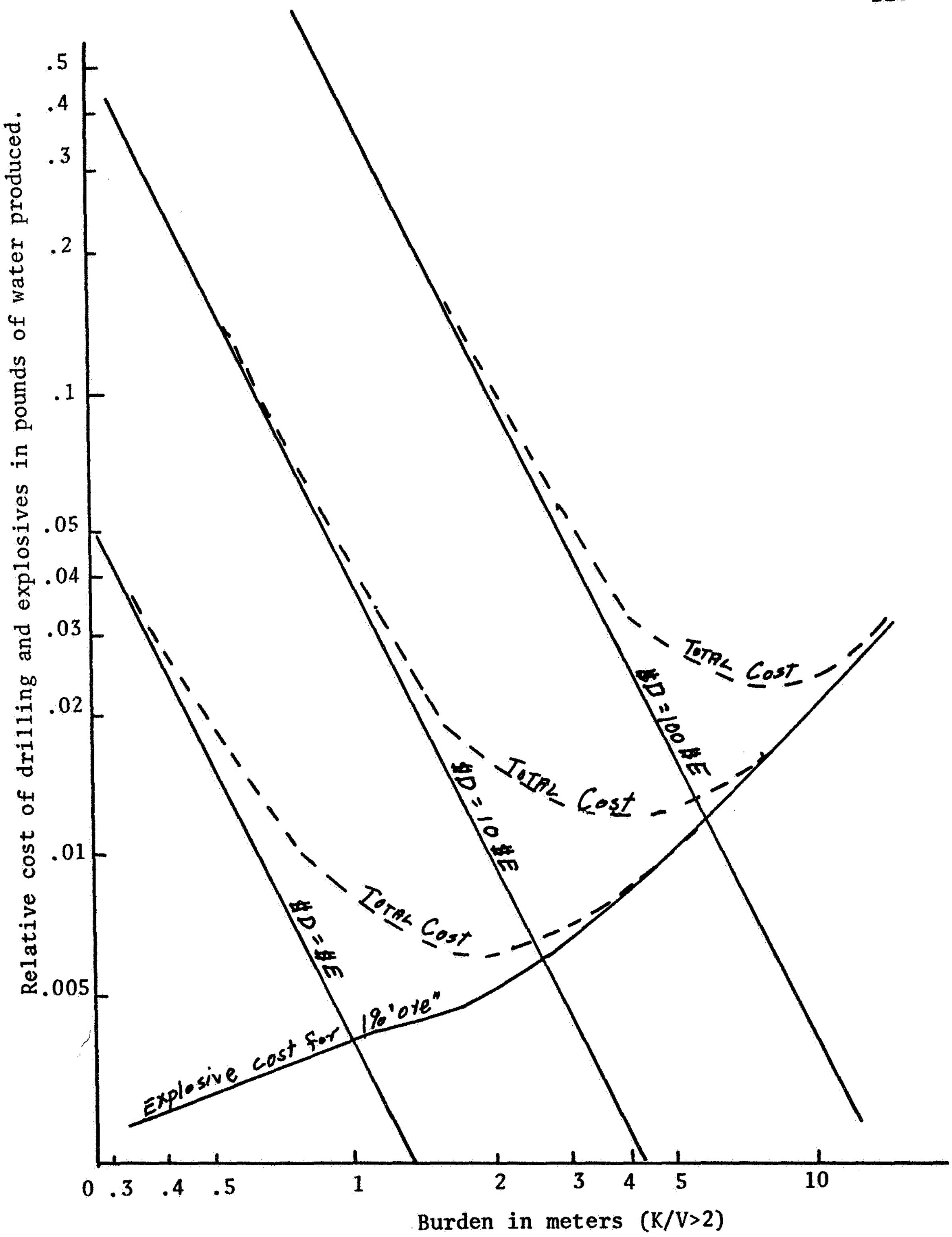


Fig. 22. Cost of fragmentation for a 1% ore in terms of pounds of water produced for various estimated drilling costs.

13. Nuclear Explosives: The application of nuclear explosives to surface mining would be limited to:

(1) stripping overburden away by cratering, 73/ and

(2) in situ fragmentation of ore and rock 72/.

The results of tests conducted in Operation Plowshare under the auspices of the Atomic Energy Commission have proven the technical feasibility of cratering and in situ fragmentation utilizing nuclear explosives. The economics remains to be proven; however, for lunar operations it may well be justified.

14. Stripping Overburden: The use of nuclear explosives to strip overburden by cratering has one big disadvantage; the ejection of large volumes of rock, and the possible radioactive contamination of the lunar environment. Also for mining purposes, the explosives would have to be placed above the ore body, and the physical dimension of a crater indicate that very little surface area of ore would be uncovered relative to the depth and volume of the crater produced. For this reason it is the authors' opinion that nuclear explosives could not be used in this manner.

15. In Situ Fragmentation: Fragmentation of an ore body and/or overburden prior to surface mining can be accomplished utilizing contained nuclear explosions without ejecting rock or contaminating the ore body or environment 74/.

Fragmentation would be performed by placing nuclear devices of a predetermined size at considerable depth below the "ore" as shown in Fig. 23. Upon detonation of the device, a cavity would be formed which within a few hours or days would collapse and proceed upward forming a zone or chimney of broken rock. By choosing the correct depth of implacement, the chimney will intersect the surface and form a slight subsidence crater and of more important a zone of fragmented "ore." This "ore" could then be mined by surface method.

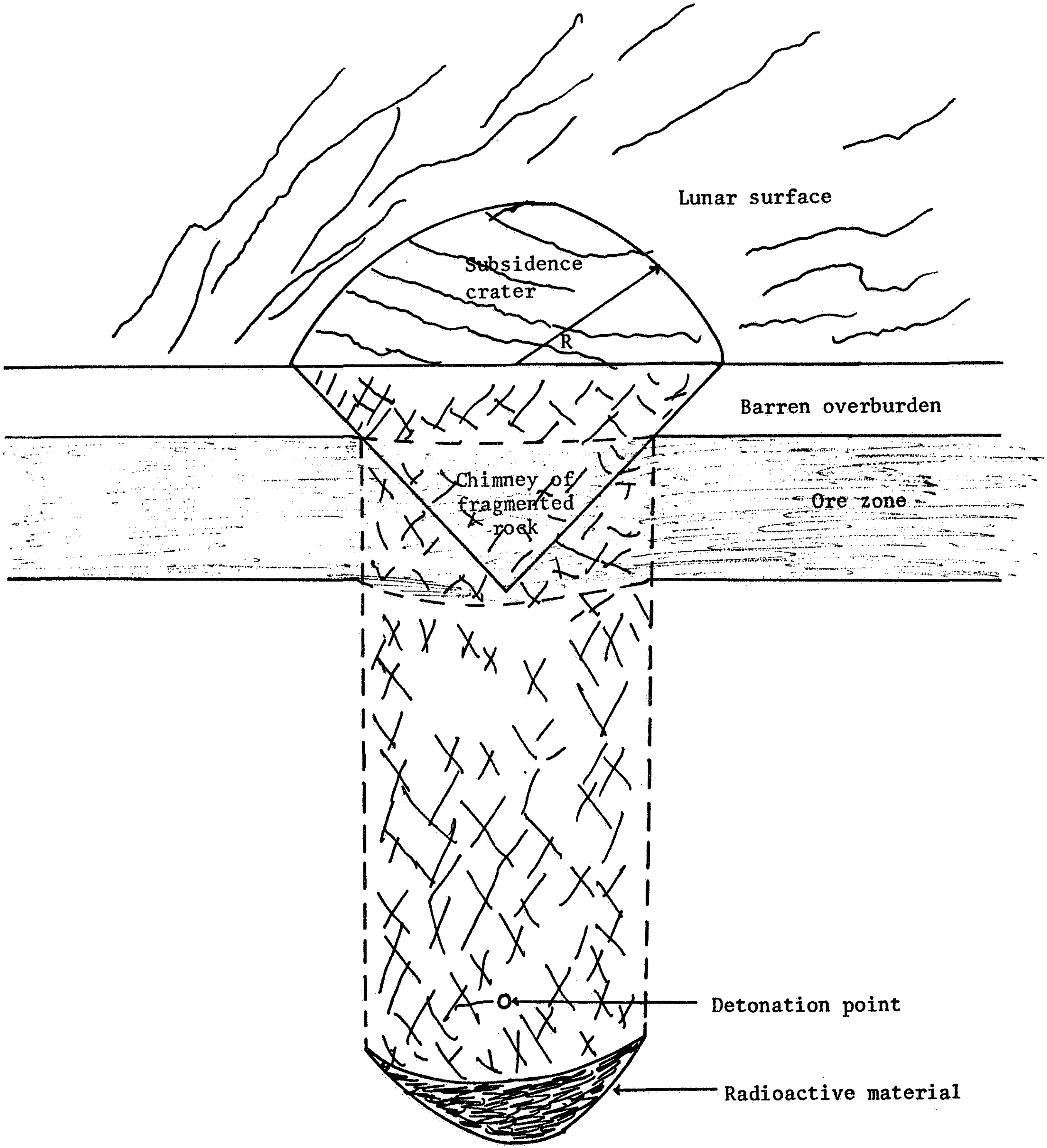


Fig. 23. **Schematic cross-section of in situ nuclear fragmentation of a lunar orebody for surface mining.**

From tests conducted by the AEC 74 /, it has been determined that the radius of the chimney can be related to the depth of burial and yield of the explosives by the equation:

$$R = C \frac{W^{1/3}}{(ph)^{1/4}} \qquad (18)$$

where R = radius of chimney, ft.
W = yield of nuclear device, Kt.
C = constant dependent on rock type
p = density of overburden
h = depth of burial of device or height of chimney, ft.

Also, the height of the chimney can be related to the radius by the equation:

$$H = kR \qquad (19)$$

where H = height of chimney, ft.
k = constant dependent on rock type

Assuming the density of lunar rock to be 1/6 that of corresponding earth rock, equation (18) can be modified for lunar applications as:

$$R = \frac{CW^{1/3}}{\left(\frac{ph}{6}\right)^{1/4}} \qquad (20)$$

Since all variables except density are constant for a given situation, equations (18) and (20) can be combined to yield the lunar charge as a function of the equivalent earth charge as:

$$W_{lunar} = .26\ W_{earth}, \text{ (R, C and h constant)} \qquad (21)$$

Values of the constants C and K for various rock types, are given in table 13 74 /.

Table 13

CONSTANTS FOR VARIOUS ROCK TYPES

Rock	Density, earth	Density moon	C*	k
Alluvium	1.6	0.27	284	8-10
Granodiorite	2.7	0.45	261	4.5
Tuff	1.8-1.9	0.31	343	5.3
Salt (open cavity produced)	2.26	0.38	381	1.6 (no chimney produced)

By selecting a desired chimney radius, dependent on mining requirements, it is possible to calculate the size nuclear explosive needed and the depth of burial necessary for the chimney to intersect the surface and produce a zone of fragmented ore.

Table 14 contains sample calculations of the size device required and the specific consumption of explosives for a lunar deposit similar to granodiorite. It is assumed that less dense deposits would require less explosives.

Table 14

Calculation of Nuclear Device for Fragmentation
Granodiorite: C = 261, k = 4.5, ρ_{earth} = 2.7

R, ft (m)	h, ft (m)	W_{earth}, Kt	W_{lunar}, Kt	Tons Broken	kg/m^3 (equivalent)	kg/kg water (1% ore assumed)
10 (3.3)	45 (14)	.0022	.0006	77	.016	.0004
100 (33)	450 (140)	12.3000	3.2000	77×10^3	.083	.0021
325 (100)	1460 (450)	965.0000	250.0000	2.76×10^6	.180	.0045

From Table 14 it can be concluded that the cost of fragmentation with nuclear explosives is approximately the same as with high explosives (see Table 12) for larger blasts and less for the smaller blasts. Also a definite advantage of nuclear explosives would be less drilling cost per unit of water plus lower explosive costs in terms of water produced per Kg. However, the depth necessary for burial will exceed 100 feet in most cases, which might present a problem for drilling.

VII

TENTATIVE ASSESSMENT OF PROBLEMS OF -

1. Overburden Disposal: If mining operations are conducted on the relatively level surface of the maria, overburden disposal should present no special problems. Large quantities of overburden, or material that adheres to bulldozer blades, buckets, scrapers, etc., would be hard to handle. In general, overburden from deposits No. 1, 2, 3, and 6 can be dragged or pushed to, hauled to and dumped in, the space left when the ore is removed. If overburden volume is very large or if the breaking of it results in a greatly reduced bulk density (increased volume/unit mass) it must be piled very high in the mined area or piled outside the pit. If the ore body has great lateral extent and mining does not begin near the edge of the deposit, some of the waste material may have to be handled again in later mining, if piled on part of the deposit. In such a case, long range mining projections should be made early in the life of the mine.

In mining deposits 4, 5 and 7, waste material must be piled on the mare surface, outside the mine. If piled on the footwall side, it would never have to be moved again no matter how deep the mine is extended. In the case of deposit No. 7, if the parallel fissures are too close to provide piling space, mining should begin on an outside fissure and move inward with refuse piled not only in the space between fissures but also in and on the mined out fissures. This practice may preclude ever going back and mining the narrow fissures to a greater depth by any method, including underground.

In the case of mining a very large crater, like the small one of Fig. 3, the rim cannot be removed and mining probably must be confined to the inside of the crater. In this case, the overburden must be piled up on the crater walls and care must be taken that it does not slide into the mine, at least until mining has proceeded a considerable distance from the crater wall.

There seems little chance to use lunar gravity to aid overburden disposal unless it can be dumped into a nearby barren rill or unless mining is high on some steep slope so that it may be dumped down the hillside. The new Kentucky strip mining law regulates such practices but there are no flowing streams to pollute or muddy on the moon!

If any material is to be removed from all or part of the overburden, it will be diverted to processing before final disposal. If not processed at the mine, final disposal will not be a part of the mining problem.

2. By-product recovery: Many substances may be inferred to be present near the lunar surface. Some of these may be useful to lunar technology, even in the primitive stages. Suggested substances are meteoritic iron, diamonds, sulfur, ferrous sulfide (troilite), olivene, basalt, halide salts, sulfates, boric oxide, graphite, many metallic sulfides, carbides, and solid hydrocarbons 3,4,5,7/. With the over-all lunar composition apparently so different from that of the earth, it is possible that surface chemistry may be different also and minerals may be present other than those found in meteorites and associated with volcanic terraines.

If some useful mineral, or minerals, occur in the material overlying lunar water deposits, it may prove economic to recover all or part of it. Such recovery probably will have little effect on the mining system or equipment used, except to divert overburden from direct and immediate disposal to some processing system. Such a system may be simple, like hand-picking or passage over a magnet, with or without prior crushing, or it may be complex like a pressurized, closed system for flotation of diamond. It is obvious that if a large volume of material must be processed for a comparatively small quantity of mineral of low value, the process must be simple and of high capacity. Any recovery of material from overburden at a water deposit, by any method other than handpicking, would actually amount to a mining operation with two or more processing plants and with both streams of material from the mine going to processing. Another possible situation would be immediate disposal of overburden but with the view of its later recovery for processing to recover minerals of secondary value. Our knowledge of lunar surface chemistry is so meager that active investigation of plans for processing lunar surface material, at present, must be limited to meteoritic iron, ferromagnesian rock (olivene and pyroxene) and possibly, troilite and sulfur.

3. Water Processing: Considerable work has been done on the heat of decomposition of certain earth hydrous minerals or other hydrous substances 75-79/ but the authors know of no work dealing with a wide range of hydrous substances and the energy demands and costs of the entire operation of dehydration of rock, electrolysis of water and storage of LOX and LH_2. One paper on energy requirements for rock dehydration, is available 80 /. Water electrolysis technology on earth is yet rather primitive and needs to be made much more efficient for lunar use 80 /. Even before this is done, however, a cost study is needed on the processing of (1) ice, (2) an ore with physically bound water like a hydrous tuff or pumice, (3) hydrated salts like $CoCl_2 \cdot 6H_2O$, $FeCl_2 \cdot 4H_2O$, $NiBr_2 \cdot 3H_2O$, $MgSO_4 \cdot 7H_2O$, etc., (4) hydroxyl-containing minerals like serpentine and (5) basic igneous rock containing 1-2% water. All steps, from delivery of the ore to the plant to the storage of LO_2 and LH_2 at the lunar space port or colony shelter, should be included. Such a study should indicate if costs of processing are likely to be comparable to, or smaller than, those indicated for mining. By including a wide range of possible ores, from ice to rock, the order and range of costs should be apparent.

All costs, from deposit exploration drilling (discovery may be chargeable to the scientific aspects of lunar exploration), through mining and processing, to product on site delivery, must be approximated before it will be known if indigenous water supplies are likely to be economically available.

4. Product Storage: Papers on lunar cryogenic storage 81, 82/ of LO_2 and LH_2 have been published and cost calculations may be possible now. Storage of supplies for mining, particularly explosives, also deserve study but in the present work it was necessary simply to assume that these could be stored and used (Sec. VI).

5. Lubrication and Maintenance: Work has been done on lubrication in vacuum 83 / and undoubtedly more is now being done. Lubrication problems of mining machinery probably will not differ from those of other equipment. On earth, maintenance of mining equipment often presents special problems of dust and mud contamination but if all lunar equipment bearings are vacuum sealed, the problem should be absent. Work has been done on motor brushes for vacuum use 58/ and it is probable that the action of other parts of moving machinery in vacuum should be studied.

Both routine and emergency maintenance probably will be done in a shelter, under earth environment conditions, unless the equipment is too large to get into the shop. When the defective part cannot be detached and carried inside, and when it is removed from and replaced on the machine, the work will have to be done outside. In the lunar daytime, awnings can be used for protection from the sun's heat and the solar wind. Operations like welding metals or splicing steel cables may present special problems when done in a vacuum. One reason for having an entire stand-by machine is to allow maintenance to be done on a non-emergency basis. If both machines are broken down at the same time - an unlikely happening - it is not probable that the defective part in each will be the same, so it will be possible to "cannibalize" each machine to fix the other until spare parts can be obtained from earth. Parts known to be subject to stress and that may fail, should certainly be kept in liberal inventory. Design of lunar equipment for mining, hauling, general transportation, etc., should be correlated so that many parts are interchangeable and parts inventories can be kept adequate, but minimum.

6. Transportation and Handling of Ores and Supplies: This topic was discussed briefly in an earlier report under this contract 1 /. It is a very important one and deserves further study. Mining operations probably will present the greatest masses to be transported and handled of any lunar operation. They will be related to general transportation problems and will involve the largest volumes of material but probably not the longest haul distances. Differences from other transportation will be similar to those on earth between passenger and freight traffic.

7. The Relation of Lunar Mining to Lunar Bases: It seems logical that mining operations will be carried on near a general lunar base. If sufficiently large, they may even form the core, or chief activity, of such a base. Earlier bases for lunar exploration, or "quasi-permanent" ones 18 /, will not necessarily be the sites of later permanent or long term bases. Because of the size of mining and processing crews, the presence of laboratory facilities, earth communication facilities, maintenance shops, it seems almost certain that the mine site, or sites, will also serve as the base for other lunar activities, such as exploration, earth weather observation, astronomical observations, etc. The necessary presence of maintenance facilities should result in mine sites being a terminal for lunar surface transportation routes. It is likely that space exploration stations, using the fuel and oxygen produced, will be located near the mining site, unless the terraine is unfavorable.

8. The Relation of Mining to Lunar Exploration: Lunar exploration will precede lunar mining by a number of years. After gaining some idea regarding general selenology, mineralogy and selenologic structure, the discovery of suitable water deposits, strategically located, may very well become a major objective of the lunar exploration program. Of course, if water deposits prove common, their discovery may be made by early exploration expeditions.

After lunar mining begins, observations and analyses made at the mine, should add greatly to the detailed knowledge of lunar mineralogy, petrology and even structure. The mine site with its shop facilities, will be the ideal location for deep exploratory drilling for scientific or other purposes. If all lunar mining operations are on the maria, however, deep drilling in the highlands area must be done elsewhere. Mines located near the edges of the maria may still serve as the base for such drilling. Observations made during mining operations have added much to our knowledge of geology, geologic processes, structure, and geologic history and we may well expect our knowledge of selenology to be expanded in the same way.

VIII

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